Economics of Underground Conversion in an Operating Limestone Mine

by

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ABSTRACT

This thesis deals with the appraisal of the economics of underground conversion in surface limestone mining operations. Software that predicts the time at which an open pit operation should be converted to underground extraction has been developed. The software is based on estimates of capital expenditures required for the underground conversion and for future equipment acquisitions and replacements under both open pit and underground operating alternatives, as well as long-term operating cost estimates for both alternatives. Open pit and underground cost estimates can be entered either directly, or estimated using O'Hara and Suboleski's (1992) cost estimation equations. It is assumed that an underground source of limestone is accessible and that its extraction is technically feasible. The program determines the cost-flow profile of each alternative and compares their present worth equivalents at yearly intervals over a pre-determined period of analysis. The program reports the optimum time for the conversion, if indeed it exists.

The report starts with a review of limestone and dolomite as mineral commodities. This is followed by a brief literature review relating to underground conversion of surface mining operations. Mining methods and costs related to industrial minerals are then described and discussed. The thesis concludes with a detailed description of the software and a hypothetical case study.

RÉSUMÉ

Ce mémoire analyse la justification économique de la conversion d'une mine de calcaire à ciel ouvert à une opération souterraine. Un logiciel informatique qui prédit le moment le plus propice à faire cette conversion a été developpé. La décision se base sur les investissements nécessaires pour effectuer la conversion ainsi que ceux relatifs à l'acquisition et au remplacement futurs d'équipement sous les deux modes d'extraction, et sur les frais d'exploitation à long terme. Il est possible à l'utilisateur de fournir ses propes coûts, ou de laisser au logiciel le soin d'utiliser les fonctions de coûts de O'Hara et Suboleski (1992). Il est supposé que la source souterraine de calcaire est accessible et qu'il est techniquement possible de l'exploiter. Le logiciel détermine le profil des coûts des alternatives et compare leurs valeurs actuelles à intervalles annuelles pendant une période d'analyse établie au préalable. Le logiciel indique le moment opportun de procéder à la conversion, si en fait il existe.

La première section de ce mémoire discute du calcaire et de la dolomie en tant que minéraux industriels. Ceci est suivi d'une brève revue de la littérature traitant de la conversion des mines à ciel ouvert en opérations souterraines. Les méthodes ainsi que les coûts d'extractions relatifs aux minéraux industriels sont ensuite décrits. Une description détaillée du logiciel et une étude de cas hypothétique complètent le document.

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Note on the Units of Measurements Used

Both S.I. and imperial units of measurements are used in this thesis. Imperial units are mainly used in chapters 4 and 5, whereas S.I. units are used in the remaining chapters. The reasons for using imperial units in specific parts of this report can be justified by the following:

- the software described herein was initially developed for a U.S.-based limestone operation which uses imperial units;

- the majority of underground limestone mines are located in the U.S., the most important market for the software;

- most available publications relating to this subject reported measurements in imperial units.

Hence, for the chapters related to the software description and the case study, i.e. chapters 4 and 5, all measurements are reported in imperial units.

A Table of Conversion of imperial units to their metric equivalents is provided below.

Table of Conversion (Imperial to Metric Units)

Imperial Units	Multiplying Factor	Metric Units
foot (ft)	0.3048	meter
inch (in)	25.4	milimeter
short ton (ton)	0.907	tonnes
cubic yard (yd3)	0.7645	cubic meter
cubic foot (ft3)	0.0283	cubic meter

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THESIS DESCRIPTION

1.1. Introduction

Although they are a major contributor to, and an indicator of the economic growth of any nation, limestone and dolomite operations usually do not draw the same level of technical attention as their metal commodity counterparts. The reason for this is the relative abundance and consequent low unit value of these commodities. This has restricted the industry to traditional surface operations, seldom inspiring operators to venture into underground mining. Such a situation, however, has changed. The dramatic increase in public awareness of environmental issues in the last 15 years, the strategic importance of site location due to increasing transportation costs, and the trend towards cost minimization, have all contributed to a changing scenario.

Because of their common location in the vicinity of urban areas, limestone and dolomite producers have been under constant pressure to comply with local, provincial, and national requirements concerning a wide range of environmental issues. Noise, dust, aesthetics, flyrock and reclamation are just a few of those environmental issues related to surface mining. Consequently, permitting and zoning have become a major obstacle for a surface limestone and dolomite producers.

Apart from the environmental implications, the progressive expansion of a surface mining operation may result in an increased stripping ratio and haulage distance, with a direct increase in operating costs. The move to an alternate location may imply costlier transportation to the captive market and often the need for a new processing plant. In those cases, underground mining becomes attractive. Weather as well may play an important role since in some places, open air extraction is limited to a certain period of the year. Post-mining use can be another advantage of underground mining in locations at which space is in high demand. Warehouses, parking garages, offices, industrial production and libraries are just a short list of potential uses for underground space in urban areas.

The latest statistics available show that there were 109 active underground limestone/dolomite operations in the U.S. in 1993. Several other producers are in various stages of feasibility analysis, planning, permitting and development. It is expected that the amount of stone produced underground will triple over the next 15 years.

Therefore, although limestone production from surface operations will undoubtely remain the principal method of mining, underground operations will become an increasingly more common method of limestone production.

1.2. Objective

In order to maximize the economic benefits of converting to an underground operation, the anticipation of the right time for the conversion is fundamental. A significant amount of lead time is required before the underground mine can start production. From the identification of the need of conversion until the beginning of construction, the process must pass through many stages: ascertion of the availability of underground reserves, preliminary feasibility study, full feasibility study, corporate approval, permitting, and bidding. The entire process can take from two to four years. Most often, however, operators are so deeply involved in short-term production problems that there is a lack of time and resources to carry out a preliminary economic evaluation, thus, missing the optimum time for underground conversion.

The development of software that enables the mine operator to determine the optimum time of conversion based on DCF techniques and O'Hara and Suboleski's cost estimation equations is the major objective of this thesis. It is assumed that the existence of underground reserves has been confirmed and that their extraction is technically feasible.

The program considers only tangible factors that can be directly expressed in monetary terms. Intangible factors, such as the social impact on the local community and the psychological impact on the workers resulting from a new underground work environment, are not taken into account.

The results of the program should be viewed only as a preliminary level indication of the economics of underground conversion. No final decision to proceed with the conversion should be taken based solely on the program's output.

The software was initially developed for use by a corporation operating in the U.S. For this reason, the imperial system of units is used in many parts of this document.

1.3. Thesis Organization

The thesis is divided into six chapters. A brief description of each follows:

Chapter 1 - Thesis Description: comprises an introduction, a statement of objectives, and the thesis structure;

Chapter 2 - Limestone and Dolomite: describes the geology, applications and market aspects of both commodities, with special attention given to their use as cement and aggregate stone;

Chapter 3 - Open Pit and Underground Mining Methods: reviews the most common limestone mining methods, their costs and respective environmental impacts. A brief review of literature addressing underground conversion is provided;

Chapter 4 - Economic Break-Even Program: describes the content of the program as well as assumptions made and algorithms used;

Chapter 5 - AS Cement Inc. Case Study: documents the process of the underground conversion analysis made in a hypothetical limestone mine;

Chapter 6 - Conclusion and Future Prospects: evaluates critically the usefulness and limitations of the program, and suggests further dev

LIMESTONE AND DOLOMITE

2.1. Introduction

Most people do not realize the importance and size of the limestone and dolomite business. During 1995, 804 million tonnes of crushed limestone and 93.1 million tonnes of dolomite were produced in the United States. In Canada, 85.7 million tonnes of limestone were produced in 1994. In terms of quantities, sand and gravel was the only mineral commodity produced in greater amount in the U.S. In 1995, limestone or dolomite was produced in 47 of the 50 states in the U. S., with Delaware, Louisiana and North Dakota being the only non-producers. Saskatchewan is the only province in Canada without limestone production.

Limestone and dolomite are high-volume, low-value commodities mostly used for construction purposes as aggregates with or without a binder. Cement and lime manufacture, agriculture, and metallurgical flux are some of the other uses of limestone and dolomite. Carbonate rocks, including limestone and dolomite, constitute about 15% of the earth's sedimentary crust and are found extensively on all continents.

The purpose of this chapter is to provide an overview of these mineral commodities before describing their extraction methods in Chapter 3. The most relevant aspects related to these mineral commodities are discussed within the next four sections. Firstly, geological aspects such as mineralogy, genesis of deposits and exploration are briefly introduced. Secondly, chemical and physical properties of limestone and dolomite are discussed. Production and uses are presented in the third section. Finally, the fourth section outlines market issues and future outlook conditions. Mining aspects and their related environmental consequences are discussed in Chapter 3.

Most of the information found in this chapter was compiled from the following sources: *Industrial Minerals and Rocks - 6th Edition*, chapter entitled *Limestone and Dolomite* (Carr et al., 1994); *Mineral Resources, Economics and the Environment*, chapters 11 and 12 (Kesler, 1994); and *The Aggregate Handbook* (Barksdale, 1991).

2.2. Geology

2.2.1. Mineralogy

Limestone and dolomite are carbonate rocks. Limestone is a sedimentary rock composed of the minerals calcite and aragonite, which have the same composition ($CaCO_3$) but different crystal structures, and the mineral dolomite [$CaMg(CO_3)_2$], with lesser amounts of chert, apatite, pyrite, hematite, and clastic sand, silt, or clay. Pure limestone is known as high-calcium limestone, and rock containing a high proportion of dolomite is known as high-magnesium dolomite. If clastic silicate impurities dominate, the rock is called marl (Carr and Rooney, 1983; Reading, 1986).

Different physical properties such as specific gravity, colour and crystal form are used to distinguish one carbonate mineral from the other. Calcite presents a hexagonal crystal structure, with good rhombohedral cleavage. Its specific gravity is 2.72 and it is commonly colourless or white, but may be other colours due to impurities. Dolomite has also a hexagonal crystal structure but its specific gravity is higher, at 2.87. Aragonite has the same chemical composition as calcite but has a orthorhombic crystal structure. Its specific gravity is 2.94. Graf and Lamar (1955) and Tucker and Wright (1990) published detailed information on the chemical, mineralogical, and physical properties of carbonate minerals.

In the field, hydrochloric acid is used to identify the different minerals. Calcite and dolomite have different rates of solubility. Calcite is more soluble in dilute acid than dolomite. Hence the amount of calcite can be estimated by the amount of dolomite left standing on a rock surface that has been exposed to dilute acid. In the laboratory, other more advanced techniques can be applied. For instance, staining of thin sections is particularly effective (Dickson, 1965, 1966). Other laboratory techniques utilize the X-ray diffractometer to determine carbonate mineralogy of bulk samples. Based on the comparison of diffraction intensities with those of known standards, it is possible to determine the amount of calcite and dolomite in a sample.

Another property of carbonate rocks, colour, must be used carefully as an indication of composition. Carbonate rocks are very susceptible to colour variations caused by very little amounts of noncarbonate material. Most high-purity limestones are shades of light brown and gray to white. In the presence of minerals containing ferrous iron oxides or carbonaceous matter, limestones acquire shades of gray or green. With an increased state of oxidation, the colour changes to yellows, browns or reds. A colour reference chart is useful in maintaining uniformity of rock descriptions.

Impurities in carbonate rock vary considerably in type and amount. The most common impurity is clay, which is basically composed of silica tetrahedra and alumina and/or magnesium octahedra. The clay minerals -- mainly kaolinite, illite, chlorite, smectite, and mixed-lattice types -- may be either disseminated throughout the rock or concentrated in laminae. Another common impurity is chert, which can be disseminated as grains throughout the rock, or concentrated in nodules, lenses, or beds. Chert is composed mainly of very fine grained quartz. As it easily absorbs impurities into its structure, it is found in many colours.

Silica is also found in carbonate rocks as discrete silt- or sand-size grains of the mineral quartz. These grains may be disseminated throughout the rock or concentrated in laminae and beds. Detrital limestone may contain a considerable amount of quartz silt and sand. A common constituent of limestones is finely disseminated organic matter which may give a pronounced brown or black colour to the rock.

In order to reveal trace amounts of a wide variety of other minerals in carbonate rocks, thin-section and insoluble-residue studies are carried out. These trace elements may affect the economic usefulness of rocks used for chemical purposes, but usually have little effect on rocks used for their physical properties.

2.2.2. Classification

Different aspects of carbonate rocks can be used as the basis of a classification scheme, but the most useful are composition and texture. Composition refers to the mineralogy, types of fossils or grains, and chemical constituents. Texture deals with both depositional and post-depositional features, such as relative proportions of framework grains and lime mud, grains size, cement, and pores.

Carbonate rocks seldom have a monomineralic composition in nature. Thus, variations in the amounts of calcite, dolomite, and non-carbonate materials must be considered for mineralogical classification. This classification is useful in rock descriptions but is not sufficient for industrial purposes, because certain uses have special chemical requirements. These are stated in terms of chemical composition rather than mineralogical composition, and specify the quantity of $CaCO_3$ (or CaO) and $MgCO_3$ (or MgO) in the rock, along with the maximum amount of impurities acceptable.

Leighton and Pendexter (1962) developed a textural classification system considering that most limestones can be characterized by the types and relative amounts of four textural components: grains, lime mud (micrite), cement, and pores. The ratio of the relative amounts of grains to micritic material is the basis of their system. Other classifications, such as the ones by Folk (1962) and Dunham (1962) make use of framework grains to mud ratios.

Dolomite may require a different treatment than limestone because a textural classification can be used only if the original depositional texture is preserved. A classification based on crystal size may be needed.

2.2.3. Genesis

Most limestones are the direct or indirect product of organic activity. Since most forms of life require light, the limestones usually form in shallow water. Some limestones consist of the skeletal remains of coral, molluscs, and algae, which form reefs and similar structures. Others are composed mainly of fine-grained material known as micrite (fecal matter), shells of small organisms, and calcite that was precipitated from seawater. In areas of wave action, known as high-energy environments, the limestones contain smaller amounts of clastic silicate sediment and sometimes oolites, which are spheres made up of concentric structures of calcite that grew over a small nucleous. In low-energy environments, such as in lagoons, limestones often contain significant proportions of silicate impurities and cannot be used for specialized markets. Limestones that form in deep water consist of the shells of small floating organisms that sink and form a carbonate mud. Calcite and aragonite dissolve in water depths over about 4 300 meters, preventing the accumulation of carbonate mud in extremely deep seawater (Blatt et al., 1980; Reading, 1986).

Dolomite is an alteration product of limestone which goes through chemical changes, known generically as diagenesis, after deposition. Dolomitization is the most important process of diagenesis in which Mg-bearing water transforms limestone to dolomite. Dolomite can also form by hydrothermal alteration, where Mg-rich hydrothermal brines invade limestones at depth. Other reactions that can affect limestones include the deposition of pyrite and silica (usually in the form of chert), both of which are undesirable impurities from a commercial point of view (Blatt et al., 1980).

Carbonate rocks comprise 15% of sedimentary rocks. Their deposition occurred from Precambrian to Holocene time and they represent only about 0.25% of the volume of the earth's crust (Parker, 1967).

2.2.4. Exploration

In North America, exploration for limestone and dolomite consists mostly of a detailed examination of known deposits. Usually, some data are already available in the published reports or files of state, provincial, and national geological surveys. Therefore, the first step in most cases begins with a search of these records to find the deposits that satisfy particular requirements. A sampling program of favourable deposits follows. Sampling is a very important step in the exploration process since it determines the validity of further study and may become the basis of the feasibility analysis. The most common sampling methods are coring, rock bitting, and surface sampling.

Coring is usually the best method of exploration because of its representativeness. It avoids contamination by soil and weathered material, and retains the surface material that may have worked down into solution cavities. Core taken on a regular grid pattern constitute a more representative and unbiased sample of a deposit than a single core. Therefore, once a suitable deposit is found, it should be drilled on a relatively regular grid. Many factors such as the homogeneity of the deposit, topography, cost of drilling and the use of the limestone must be assessed before determining the best drilling method and grid.

In terms of reliability, drill cuttings are probably the least reliable samples in exploration. In order to improve reliability, drilling has to be done in a carefully cased hole to prevent overburden contamination, and the cuttings must be collected carefully. In addition to that, an experienced geologist in drill cuttings is also required for the interpretation.

Chip samples can provide a good representation of the deposit if taken carefully. They should be collected from unweathered surfaces even if the weathered rind has to be chipped away. Samples should be washed to remove contaminants, but care should be taken not to wash out thin, interbedded shales.

2.3. Properties of Limestone and Dolomite

2.3.1. Physical Properties

The physical properties of limestone and dolomite determine whether they are suitable or not for specific applications such as construction materials. Different physical tests exist to verify the suitability of a rock for a particular use. The American Society for Testing and Materials (ASTM) and the American Association of State Highway and Transportation Officials (AASHTO) provide explicit procedures for physical testing of limestone and dolomite. Some of the physical properties that can be tested include apparent specific gravity, apparent porosity, compressive strength, modulus of impact rupture, toughness, tensile strength, abrasive hardness, Young's modulus, modulus of rigidity and Poisson's ratio. The now defunct USBM has undertaken considerable research on the properties of rock, many of which have applications in blasting and mining (Atchison et al., 1964; Atchison and Pugliese, 1964; Bur et al., 1969; Dick et al., 1973; Merril, 1956; Pugliese, 1972; Thill et al., 1969; Willard and McWilliams, 1969).

Physical tests of carbonate rock used for aggregate have been carried out in connection with state and federal road building programs. However, much of this information is unpublished. Research related to the relationships of the physical properties of carbonate rock to its use as an aggregate has been sponsored by the Highway Research Board of the National Research Council. Renninger and Nichols (1970) reviewed some of this work and gave the current status of aggregate research. A useful overview of sampling and testing principles is given by Marek (1991).

Another source of information regarding physical properties of carbonate rocks is the U.S. Army Corps of Engineers, which has tested samples from many quarries in the United States. Their purpose was to assess potential uses in constructing locks, dams, and other structures.

One physical property that concerns the construction industry is rippability. Limestones that are thinly bedded, low in compressive strength, or sufficiently inhomogeneous can generally be ripped easily. Caterpillar Tractor Co., through empirical testing, found in 1972 that the higher the wave velocity, the more difficult it is to rip the rock (Caterpillar Tractor Co., 1972).

2.3.2. Chemical Properties

The chemical and physical properties of carbonate rocks are interrelated. As an example, pure calcite in the form of poorly cemented chalk presents low strength and it is very reactive. However, pure calcitic marble of the same chemical composition is relatively strong and unreactive. Mineralogical composition also plays an important role. Dolomite containing quartz sand grains may have the same composition as dolomite with chert, but their suitability as aggregate differs due to the difference in reactivity of the two forms of silica. Therefore, physical and mineralogical descriptions of carbonate rocks are useful in determining the chemical properties of the product from a deposit.

Chemical determinations are essential for some uses of carbonate rocks such as glass raw material, flux, or cement. In these cases, the amount of certain elements must fall within specified limits or ranges. In the cement industry, for instance, an MgO content above about 5% is not allowed. For stone that is used because of its physical properties, the chemical content can be sometimes important. The proportions of alumina (Al_2O_3) and silica (SiO_2) may be helpful in determining the value of a carbonate rock for a use in which physical properties are important. The higher the alumina content, the more argillaceous the rock is likely to be. For agricultural limestone, the chemical analysis is important for estimating its neutralizing value.

Most sedimentary carbonate rocks vary in their impurities since they were deposited in different environments. Therefore, in order to assess the approximate composition of a particular unit of rock, analyses of many samples are required.

Chemical data on carbonate rocks are found at the state geological surveys in the United States or at the Geological Survey of Canada.

2.4. Production and Uses

In the United States, 910 million tonnes of carbonate rock were sold or used in 1995, as shown in Table 2.4.1. Limestone and dolomite represented 98% of that total. According to the USGS, "sold or used" means the amount of production released for domestic consumption or export in a given year. Stockpiled production is not included in the reported figures. Only Delaware, Louisiana and North Dakota did not report any limestone or dolomite production. The top eight limestone and dolomite producing states in descending order were Texas, Florida, Missouri, Pennsylvania, Illinois, Ohio, Kentucky and Indiana. These states represented 55% of the total U.S. production. Table 2.4.2. lists the top eight states along with their production in 1995.

Table 2.4.1. - Carbonate Rocks Sold or Used in the United States. 1995

(modified from Tepordei. 1996)

Type	Number of Quarries	Quantity (thousand tonnes)
Limestone	2010	804000
Dolomite	182	93100
Marble	42	5960
Calcareous marl	14	4570
Shell	11	2320
Total	2259	909950

Table 2.4.2. - Top Eight Limestone and Dolomite Producing States in the U.S., 1995 (thousand tonnes)

(modified from Tepordei, 1996)

State	Aggregate	Cement	Agriculture	Lime	Others	Total
Texas	53002	9280	590	893	12400	76165
Florida	36312	W	799	-	28700	65811
Missouri	30950	5830	1220	W	27600	65600
Pennsylvania	35417	6450	673	679	18800	62019
Illinois	31900	3530	2690	-	23300	61420
Ohio	31970	1360	925	W	26500	60755
Kentucky	28470	W	1060	W	26000	55530
Indiana	23470	2510	1520	W	21700	49200

W: Withheld to avoid disclosing company proprietary data

During 1994 in Canada, limestone and marble production was 86.5 million tonnes. Table 2.4.3. presents the production figures for carbonate rock excluding stone used in the Canadian cement and lime industries, which totaled 15.4 million tonnes in 1994. Ontario, followed by Quebec, were the two largest limestone producing provinces in the country.

Table 2.4.3. - Production of Limestone and Marble in Canada, 1994 (thousand tonnes)

(Natural Resources Canada - Vagt (1995); Statistics Canada)

By Province	Limestone	Marble	Total
Newfoundland	1184	-	1184
Nova Scotia	215	-	215
New Brunswick	564	-	564
Quebec	24832	467	25299
Ontario	37521	340	37861
Manitoba	2556	-	2556
Alberta	317	-	317
British Columbia	3015	-	3015
Northwest Territories and Yukon	102	-	102
Total	70306	807	71113

Data exclude stone used in the cement and lime industries

Each limestone use requires either different physical or chemical specifications, and often both. For construction aggregate, physical specifications, such as durability and gradation are important. On the other hand, for the production of cement or lime, chemical properties are more relevant. Some specifications may be required by a particular industry and require testing properties based upon an industry standard or procedure. Sometimes, these requirements may be related to a specific producer and be unique to the specific application.

Whereas there are hundreds of applications, the most important markets for limestone and dolomite in the United States can be divided into nine categories and 38 primary uses as shown in Table 2.4.4.

Of the 804.3 million tonnes of crushed limestone sold or used in the United States in 1995, 301.9 million tonnes or 37.5% was for "Unspecified uses - actual and estimated". Of the remaining 502.3 million tonnes of crushed limestone reported by the producers, 75.8% was used for construction aggregate, 19.9% for chemical and metallurgical purposes, including cement and lime manufacturing, 2.5% for agricultural purposes, and 1.8% for special uses and products.

In terms of crushed dolomite, of the 93.1 million tonnes sold or used in the United States in 1995, 25.8 million tonnes or 27.7% was for "Unspecified uses - actual and estimated." Of the remaining, 90% was used for construction aggregate, 5% for chemical and metallurgical purposes, 3% for agricultural purposes, and 2% for special and miscellaneous products.

Table 2.4.4. - Crushed Limestone and Dolomite Sold or Used by Producers in the United States, 1995

(thousand tonnes)

(Modified from Tepordei, 1996)

Use	Limestone	Dolomite
Coarse aggregate (+1 1/2 inch):		
Macadam	2360	797
Riprap and jetty stone	16200	1060
Filter stone	4310	115
Other coarse aggregate	4110	588
Coarse aggregate, graded:		
Concrete aggregate, coarse	64600	8310
Bituminous aggregate, coarse	49200	7770
Bituminous surface-treatment aggregate	10500	2300
Railroad ballast	3670	1990
Other coarse aggregate	10600	6170
Fine aggregate (-3/8 inch):		
Stone sand, concrete	7320	1320
Stone sand, bituminous mix or seal	13800	2060
Screening, undesignated	19100	3000
Other fine aggregate	3790	1370
Coarse and fine aggregate:		
Graded road base or subbase	108000	1180
Unpaved road surfacing	18300	5870
Terrazzo and exposed aggregate	1010	101
Crusher run, fill or waste	27600	2270
Other coarse and fine aggregate	11300	3390
Roofing granules	158	-
Other construction materials	5000	229
Agricultural uses:		
Agricultural limestone	10800	2000
Poultry grit and mineral food	1390	W
Other agricultural uses	531	52
Chemical and metallurgical uses:		
Cement manufacture	79900	W
Lime manufacture	12600	1160
Dead-burned dolomite manufacture	W	W
Flux stone	3580	2060
Chemical stone	942	-
Glass manufacture	598	156
Sulfur oxide removal	2440	-
Special uses:		
Mine dusting or acid water treatment	526	W
Asphalt fillers or extenders	1500	593
Whiting or whiting substitute	1010	W
Other filters or extenders	2550	601
Other miscellaneous uses:		
Sugar refining	W	-
Other specified uses not listed	3140	133
Unspecified uses:		
Actual	186000	1990
Estimated	116000	5950
Total	804435	9311

W: Withheld to avoid disclosing company proprietary data

In Canada, Natural Resources Canada divided the 1994 national market for limestone into five categories and 31 primary uses as shown in Table 2.4.5.

Of the 85.7 million tonnes of crushed limestone produced in 1994, 19.5 million tonnes or 23% was used for chemical and metallurgical purposes, including cement and lime manufacturing. Of the remaining 66.2 million tonnes, 62.8 million tonnes or 73% was used for construction aggregate.

Table 2.4.5. - Production of Limestone and Marble in Canada by Use. 1994 (thousand tonnes)
(Natural Resources Canada - Vagt (1995); Statistics Canada)

Use	Limestone	Marble
Dimensional stone		
Rough	63	9
Monumental and ornamental stone	8	
Other (flagstone, curbstone, etc.)	51	
Lining, open-hearth furnaces	-	
Chemical and metallurgical		
Cement plants, Canada	13039	
Cement plants, foreign	1654	
Flux in iron and steel furnaces	190	
Flux in nonferrous smelters	154	
Glass factories	146	12
Lime plants, Canada	2367	
Lime plants, foreign	1124	
Pulp and paper mills	234	
Sugar refineries	16	
Other chemical uses	587	
Pulverized stone		
Whiting (substitute)	41	
Asphalt filler	85	
Dusting, coal mines	57	
Agricultural purposes and fertilizer plants	963	39
Other uses	739	387
Miscellaneous stone		
Manufacture of artificial stone	19	
Roofing granules	145	
Poultry grit	56	
Stucco dash	22	2
Rock wool	4	
Rubble and riprap	499	
Other uses	631	3
Crushed stone for		
Concrete agregate	7276	159
Asphalt aggregate	5453	1
Road metal	28393	13
Railroad ballast	730	
Other uses	20966	182
Total	85712	807

Data include stone used in the cement and lime industries.

As shown in Tables 2.4.4. and 2.4.5., cement manufacture and aggregate are the two major uses for limestone. A brief description of these uses follows.

2.4.1. Cement

Cement production is a world-class, high-tech, multi-billion dollar business. In 1994, world production was 1.37 billion tonnes. In the United States, production was 74.3 million tonnes with a total value of about U.S.$4.5 billion (average mill value of $61.07 per tonne). The leading producing countries in the world in 1994 were China with 29% of production, Japan with 7%, and the United States with 6%. Canada's production represented less than 1% of world production in 1994, with 10.6 million tonnes.

By definition, cement is a powder that is produced from a burned mixture of mainly clay and limestone. Since cement hardens by reacting with water, it is used in mortar and concrete. Hydraulic cements are those that do not only harden by reacting with water, but also form a water-resistant product.

Portland cement in its various forms is used for making structural concrete nowadays. ASTM C150 defines portland cement as a hydraulic cement produced by pulverizing clinker consisting essentially of hydraulic calcium silicates, usually containing one or more forms of calcium sulfate as an interground addition. Clinkers are 5- to 25-mm-diameter nodules of a sintered material which is produced when a raw mixture of predetermined composition is heated to high temperatures.

Limestone is the major raw material for cement production. Its chemical composition is very important. Certain relatively common elements and minerals can affect the final quality of the cement. The most problematic is magnesium, usually in the form of dolomite, which is not permitted in concentrations above about 5% MgO. Iron, if present as pyrite, can also be a problem.

2.4.2. Aggregate

Construction aggregate is fragmental rock and mineral material that can be used either alone as fill or in combined form with concrete, asphalt, and plaster. The production of construction aggregate depends on economic activity.

Public works (highways, dams, airports and others) and private construction (residential and non-residential construction) are the two major markets for aggregate producers.

Whereas the favoured type of aggregate is sand and gravel (Davis and Tepordei, 1985), crushed stone produced from massive rock is also used as aggregate. Limestone is the best rock for this purpose because it is relatively soft and easily mined. About 70% of crushed stone production comes from limestone. Harder granite and basalt supply another 20%, and sandstone, quartzite, and other materials make up the rest. Although crushed stone requires more processing and is about 20% more expensive to produce, it is used for slightly more than half of the U.S. aggregate production because sand and gravel deposits are scarce in many areas (Langer, 1988).

Not all massive rock deposits meet construction aggregate specifications (Tepordei, 1985). The more important specifications are resistance to abrasion, chemical attack, and splitting due to freezing water. Therefore, many massive rock deposits cannot be used.

In the United States, the total value of aggregate produced from limestone was about U.S.$ 1.8 billion in 1995. In Canada, this value was about U.S.$ 237 million in 1994.

2.5. Market and Future Outlook

2.5.1. Market

In this high-volume, low unit value commodity business, transportation plays a very important role in the delivered market price. In many situations, the cost of transportation equals or exceeds the FOB plant value of the stone. Therefore, due to the significant transportation cost and the large quantities of material needed, most of the limestone and dolomite is generally marketed locally.

Markets are also heavily dependent on population density and the resulting demand for building and highway construction. Therefore, there is a wide dispersion of quarries located near highly populated areas. However, local environmental concerns and increasing land values are forcing quarries to move farther from the customers, thereby increasing the price. Zoning regulations can occasionally result in local shortages of material. The location of production is also affected by the availability of different types of transportation. Producers located on major waterways and railroads, or within easy access to highways, can usually be more competitive in distant markets. This is the case of crushed stone producers in Newfoundland and Scotland, whose products are competitive in the major metropolitan coastal markets in the northeastern U.S.

In terms of transportation modes, truck haulage is most common because of its flexibility. In 1995, truck haulage accounted for 73% of the reported crushed stone produced in the United States (Garrett, 1996). Trucks are the main choice for distances up to 50 km. Trucking rates for industrial and agricultural limestone tend to be higher than those for aggregate limestone because of the specialized nature of the equipment used.

Railroads is a major player in the transportation of limestone and dolomite products. Rail haulage is often used for intermediate distances of 50 to 100 km, when producers have access to rail connections. It is not as flexible as trucking for delivery schedules. In the United States, only 6% of the reported crushed stone produced in 1995 was transported by rail.

Water transport is becoming more common for very long haulage distances where water access is available. In the United States in 1995, 7.2% of the reported crushed stone produced used water transport. Shipping by barge is relatively cheap when compared to truck and rail. According to a comparison compiled by Vulcan Materials in 1995, transport by barge costs, on average, 1 cent per ton mile, as compared to transport by rail, at 5 cents per ton mile, and transport by truck, at 10 cents per ton mile.

Another aspect of the market is competition. Most of the major population centres contain many independent producers and prices reflect the circumstances. Obviously, markets with only one producer generally have higher prices. Potential competitors are routinely taken into consideration by producers when establishing their prices.

2.5.2. Future Outlook

As stated earlier, the demand for limestone and dolomite products is heavily associated with residential and nonresidential construction levels in North America. In the United States, gradual increases in demand can be anticipated based on the volume of infrastructure construction that is being financed by the Intermodal Surface Transportation Efficiency Act of 1991, the National Highway System Designation Act of 1995, and the U.S. economy in general. In Canada, the trend towards lower interest rates and a moderate increase in construction activity are likely to support a slow but steady growth in demand for limestone and dolomite products. However, problems of urban encroachment and environmental reclamation on existing deposits are likely to continue, increasing hauling distances and delivery costs in many urban markets.

OPEN PIT VERSUS UNDERGROUND MINING

3.1. Introduction

In order to extract minerals found in the ground, it is often necessary to fragment, load, transport and crush the host rock ore before processing takes place. Such components are part of a mining operation which can be either open pit or underground. The decision between open pit and underground methods rests on different factors, such as the geometry and attitude of the deposit, rock conditions, capital and operating costs, environmental aspects and labour safety. In most cases, the decision is fairly clear. Under the same conditions, surface mining is regarded as the first choice because it is normally less expensive than underground mining. Some of its advantages are:

- allows the use of large equipment and therefore, high production;
- requires less upfront capital expenditures and operating costs;
- enables high ore recovery;
- allows for a safer and more flexible operation.

However, underground mining also offers some advantageous factors, such as:

- permits selective mining;
- enables all-weather mining and uniform production;
- is less conspicuous environmentally and requires less land.

When the decision is not clear, there is a need for a detailed feasibility study before opting for one or the other method.

There are also many deposits which can be mined by both methods, first by surface mining, and then, as the pit deepens, by underground methods. Some deposits are even exploited simultaneously by both methods, normally during the transition period between open pit to underground.

According to the U.S. Geological Survey, total U.S. mining of nonfuel mineral materials amounted to 5.4 billion tonnes in 1995. These materials included 3.4 billion tonnes of crude ore mined and 2.0 billion tonnes of mine waste and ore from development. Overall, 97% of nonfuel mineral mining was performed at surface, with the remaining 3% underground. For crushed stone, total surface mining amounted to 1.29 billion tonnes. Crude ore mined at surface operations was 1.2 billion tonnes, and the remaining 96 million tonnes was waste. Underground mining for crushed stone amounted to 45 million tonnes, 97% of which was crude ore.

In 1993 there were 2 250 limestone and dolomite open pit operations in the U.S. and 109 active crushed stone underground operations (Tepordei, 1994). The underground mines were located in the Midwest, primarily in Kentucky, Iowa, Illinois, Missouri, Indiana and Tennessee. Figure 3.1.1. shows the location of the underground operations in the U.S.

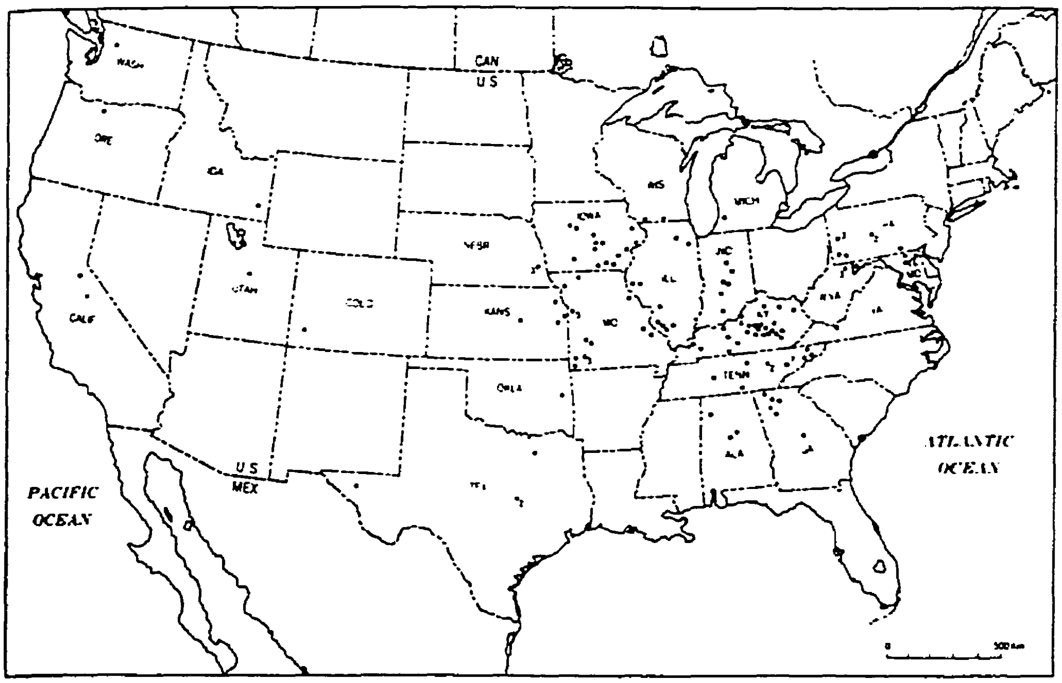

Figure 3.1.1. - Location of Active Underground Limestone Mines in the U.S. (USBM, 1991)

In Canada, total mining of nonfuel mineral materials amounted to 515.21 million tonnes in 1993. Crude ore amounted to 300 million tonnes with the remaining 215.21 million tonnes being waste. Of the 300 million tonnes of crude ore, 97 million tonnes were extracted by underground methods. In the nonmetals sector, crude ore amounted to 64.14 million tonnes, with 67% produced from underground methods.

This chapter deals mainly with open pit and underground mining methods used by the limestone industry. Three sections follow: Open Pit Mining, Underground Mining and Conversion from Open Pit to Underground Mining. In the first two sections, brief descriptions of the methods and costs related to limestone mining are presented, followed by some relevant environmental issues. In the third section, a selection of literature related to the conversion of open pit to underground mining is reviewed.

3.2. Open Pit Mining

3.2.1. Description

After a deposit has been delineated and a surface mining operation is decided upon, short and long range planning are carried out. The long term plan covers at least five years and, in many cases, up to 20 years. Corporate strategy and philosophy are often a strong input while developing the long term plan. Also at this stage, general technical parameters must be defined. These comprise items such as production rate, bench height, pit slopes, road grade, blast spacing, and equipment requirements.

For limestone, the production rate decision can be based on a market analysis for the specific application as well as on corporate strategy. However, it is uncommon to have annual production rates above 2 million tons, with the typical operation producing around 450 000 tons per year. Bench height should be set as high as possible, within the limits of the size and type of equipment selected. The bench height in limestone open pits typically averages 30 ft. The pit slope affects very much the size and shape of the pit and determines the amount of waste that must be moved to expose the ore. Geotechnical tests should be carried out to assess the proper pit slope, which can vary for different parts and depths of the mine. The addition of an access ramp can significantly affect the slope angle and therefore increase the amount of waste to be removed. The width and grade of the ramp are dependent on the truck specifications.The general rule of thumb for a two-way ramp is that its width has to be at least four times the truck width. A number of open pits have a 10% ramp grade, but an 8% grade provides more latitude in construction and in fitting the bench entries. The equipment size is a function of the productive capacity of the operation.

The long term plan also provides essential information for mine reclamation which is required by the environmental agencies. Many factors affecting the operation can change over time and therefore, an update of the long term plan is required every 3 to 5 years.

Contrary to the long term plan, the short term plan addresses the decisions to be made within a one-year period in much more detail. It includes items such as new ramp location, amount and location of overburden removal, and testing of new types of equipment and supplies, among others. Needless to say, the short term plan must comply with the long term plan objectives.

The actual development of a mine usually starts with the removal of topsoil and overburden. Topsoil is often stored separately to be used during reclamation later on. Unconsolidated to semi-consolidated overburden can be removed with basically any type of excavator such as front-end loaders and shovels. Occasionally, rippers are required to loosen the material for the excavators. In the case of consolidated overburden, however, drilling and blasting are required. Special attention must be given to the disposal of overburden in order to conciliate environmental and economic concerns.

After the pre-production stripping is completed, the primary haul road construction must be started. The haul road provides access to the production face where the extraction sequence begins. The haul road location should satisfy long term and short term requirements in the most cost effective manner. In the long term, the road should be located so that it will not have to be moved later, as the pit expands. At the same time, in the short term, it should access the production faces following the shortest route.

At the working face, drilling and blasting take place as the first productive activities. Several factors govern the type of drill to be used, such as deposit geology and thickness, rock hardness, climatic conditions, and regulatory, geographical, and local restrictions. In terms of drill hole sizes, they are typically 3 to 7 in. in diameter. The most widely used explosive product in limestone mines is a mixture of fuel oil (FO) and Ammonium Nitrate (AN). When combined in precise proportions ANFO is formed. Field experimentation is important to determine the most cost-effective choice of hole diameter and blasting agent. Additionally, drill and explosive manufacturers can offer their expertise as well as computer programs to optimize the drilling and blasting processes.

After blasting, the fragmented material is loaded and hauled to the crusher.
A typical aggregate operation of 450 000 tons per year normally utilizes off-highway trucks with capacities in the range of 35 to 85 tons. Smaller operations usually employ trucks with capacities in the 15 - 20 ton range. A majority of operations use rubber-tired front end loaders with capacities in the 6 - 13 yd^3 range, with power shovels used occasionally. Loading and hauling must be analysed as a single activity since both equipment types have to match. Some of the factors that influence the number of hauling and loading units are the haul road configuration, the possibility of relocating the primary crusher, and optimum cycle time.

Crushing is required to reduce the size of the rock before further processing takes place. For primary crushing, gyratory or jaw crushers are used. Jaw crushers have a lower capacity than gyratory crushers, but also a lower cost. Both options should be analysed technically and economically before choosing.

3.2.2. Mining Costs

Mutmansky et al. (1992) carried out a cost comparison among different commodities and mine sizes. Two surveys of mining companies were conducted by the authors to provide working data. The first survey was oriented towards coal mining; the second was oriented towards noncoal minerals. Data for the industrial mineral open pit mines are listed in Table 3.2.1. Medium-size mines are defined as those with volumes of ore and waste combined between 2 and 20 million tons per year. Small-size mines are those with annual volumes lower than 2 million tons.

Table 3.2.1. - Industrial Mineral Open Pit Mine Cost Data (Mid-1989 $)
(Modified from Mutmansky et al., 1992)

Mine	Size	Cost Estimates (US$/ton moved)					
		Drilling Cost	Blasting Cost	Loading Cost	Haulage Cost	Auxiliary Cost *	Direct Mining **
G	Medium	0.14	0.22	0.19	0.28	0.29	1.13
J	Medium	0.18	0.16	0.10	0.74	0.00	1.19
K	Medium	0.04	0.04	0.34	0.40	0.00	0.82
N	Small	0.11	0.33	0.08	2.47	0.00	3.00
O	Small	0.12	0.27	0.22	0.88	0.00	1.49
P	Small	0.09	0.17	0.17	0.92	0.00	1.35
Q	Small	0.24	0.18	0.31	0.45	0.05	1.23
R	Small	0.10	0.21	0.25	0.81	0.12	1.49
T	Small	0.19	0.18	0.43	0.00	0.70	1.50
Average		0.14	0.20	0.23	0.77	0.13	1.47
% of Direct Mining		9	13	16	53	9	

* Cost of support, pumping, roads, etc.

** Sum of drilling, blasting, loading, haulage and auxiliary costs.

More than half of the direct mining cost is attributed to haulage. Blasting and loading costs represent 13% and 16% of the direct mining cost, respectively. Drilling and auxiliary cost each represent only 9% of the total direct cost.

Some conclusions were drawn by Mutmansky et al. from the study of open pit mining costs. First, when compared to open pit mining costs in Michaelson and Hammes (1968), it can be seen that the costs have not risen rapidly over the past decades. Reasons suggested for this are the intense foreign competition and the continued increases in equipment size and efficiency. Second, the costs decrease as the size of the mines increase, probably a consequence of the higher efficiency of larger equipment, though other factors may play a part as well.

3.2.3. Environmental Issues

Environmental public awareness in North America has increased rapidly in the last 15 years and, as a consequence, mines have been forced to operate under much stricter conditions. This is particularly true for limestone and dolomite mines, which generally, must be located close to urban areas to remain competitive in the market place. Also, open pit mines are much more conspicuous environmentally than underground mines because of the very nature of their operation.

Two levels of environmental concern exist: local community sensitivities, and state and federal mandates. At the local level, the limestone industry faces zoning and community relations requirements. A good public relations program is critical, providing for the prompt and effective response to any complaints. Site appearance, blasting control, dust control, noise control, and traffic control are some of the items that must be addressed in a plan of operation in order to project a positive image. The development and proper implementation of these plans can dictate the survival of each individual mine.

At the state and federal levels, reclamation plans may be a critical part of the permitting process. When mining is complete, the top soil is usually replaced to help in restoring the vegetation, making these areas conducive to the development of parks, golf courses and housing developments, as well as promoting other uses. The pit can also be allowed to fill with rain water and natural ground water, thereby creating a lake. Some mined-out pits can be even converted into sanitary landfills, but contamination of the ground water is a major concern.

Storm-water runoff is another environmental concern under federal mandate, which requires storm-water discharge permits for all of the mining industry.

The overall response of the stone industry in the U.S. has been very positive, with the National Stone Association working extensively to promote and assist with compliance with environmental requirements.

In April 1993, *Pit & Quarry* Magazine, during one of its seminars, carried out a poll among attendees about their environmental problems and concerns. These statistics reflect the priorities of the attendees from large to small companies and may not necessarily represent the priorities of the entire industry. The attendees were asked to rank their current perceived top 10 environmental problems, as well as those in the short-term future. Table 3.2.2. shows the poll results ranked according to priority.

Of the companies represented, 76 % have written environmental policies, 80% regularly conduct environmental audits and 48 % use third parties to conduct those audits; 58 % have programs to communicate with regulating agencies.

Table 3.2.2. - Ranking of Top 10 Environmental Problems and Concerns (Weaver - 1993)

Now (1993)	*Short-term future (1995)*
1. Air quality	1. Air quality
2. Water quality	2. Water quality
3. Storm water	3. Cost of compliance
4. Wetlands	4. Wetlands
5. Waste management	5. Waste management
6. UST/AST compliance	6. Storm water
7. Cost of compliance	7. Endangered species
8. Noise	8. Reclamation
9. Reclamation	9. Noise
10. Endangered species	10. UST/AST compliance

3.3. Underground Mining

3.3.1. Description

When considering the development of an underground mine, there are several factors that should be assessed. The most important of all is the stability of the ground overlying the deposit. Therefore, rock mechanic studies are mandatory in order to determine the type of development sequence and mining method to use. Delineation of the deposit (dip, strike, thickness); selection of support methods (natural or artificial); selection of a ventilation system; presence of underground water; and means of access to the deposit (ramp or shaft), are some other important factors to be considered.

Worldwide, two underground methods are commonly applied in the limestone industry, depending on the shape and dip of the deposit. Room and pillar is suitable for horizontal or near horizontal deposits, whereas sublevel stoping is suitable for large, steeply dipping deposits. A brief description of both methods follows.

3.3.2. Room and Pillar

The room and pillar method is a type of open stoping method used in near horizontal deposits hosted in reasonably competent rock, where the roof is supported primarily by pillars. Some form of room and pillar mining is employed in the majority of underground crushed stone mines in the U.S.

The mining sequence consists of excavating several wide, straight and parallel openings with regularly-spaced, perpendicular interconnections. The pillars are formed by the unmined areas between the openings, and provide permanent support for the overlying material. Pillar dimensions and openings are determined from rock mechanics principles.

Depending on the thickness of the deposit, a single lift (pass) or mutiple-lift operation can be performed. In the latter case, the top bench is usually excavated for a short distance, then subsequent benches are started at lower levels and the sequence goes on. Each of the upper benches is kept ahead of the lower benches to provide adequate working space. The rock excavated to form the openings between pillars becomes mine production. Because the pillars are left in place, the ore recovery varies from 50 to 80 percent. Figure 3.3.1. shows a schematic view of a multiple-lift room and pillar mining sequence.

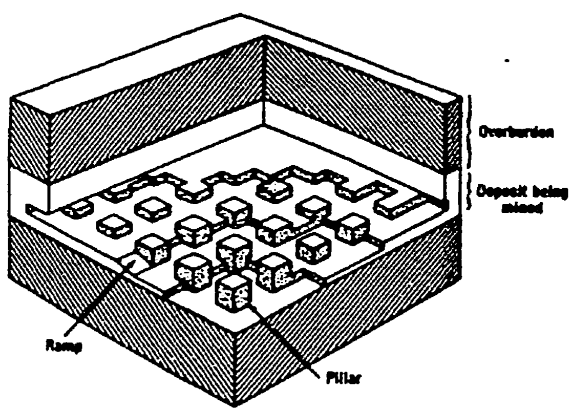

Figure 3.3.1. - Schematic View of a Room and Pillar Mining Sequence (Archibald, 1991)

In a conventional room and pillar approach, a standard drill jumbo is used to perforate horizontally from the floor when the thickness of the deposit is less than 25 ft. Extraction in this case is accomplished in one pass, which is referred to as a lift. However, if the thickness of the deposit is greater than 25 ft, the initial pass is drilled with a jumbo, and subsequent benches are perforated using surface mining-type drills. When the deposit is thick, open pit equipment can be used with minor modifications. Figure 3.3.2. illustrates an underground bench mining operation. The procedure used to select blasting agents and drilling equipment is similar to that described for open pit mining.

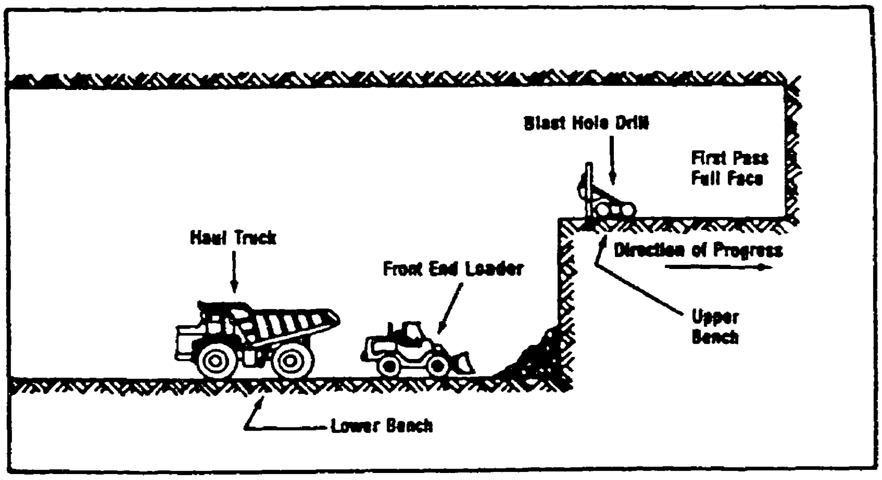

Figure 3.3.2. - Underground Bench Mining Operation (Archibald, 1991)

Due to space limitations, a large time is required to load trucks underground. Also, narrow haulage ramps allow only one truck at a time, thereby increasing waiting time. The result is an increase in the amount of equipment required. Due to their inefficient combustion and noxious fumes, regulations do not allow gasoline engines, but diesel engines are permitted. The ventilation system must have sufficient capacity to remove the hazardous gases generated by the engines.

One alternative to hoisting the ore to surface by shaft is to haul the material to surface using a conveyor system which, in conjunction with an underground primary crusher, can eliminate problems with ventilation and space restrictions.

3.3.3. Sublevel Stoping

Sublevel stoping, also known as blasthole or longhole stoping, is an open stoping, high-production, bulk mining method applicable to large, steeply dipping, regular ore bodies having competent ore and wall-rock that require little or no support. Underground limestone mines in Brazil (Benedetti et al., 1987) and Finland (Matikainen, 1981) apply this mining method. It is development intensive, although most of the development work is done in ore.

Mine development starts from a shaft sunk in the footwall to avoid possible caving effects from the stopes. Then, the ore body is divided into levels by driving crosscuts and haulage drifts every 150 to 400 vertical ft. Access raises are used to further divide the ore body into blocks for stoping. Stoping is accomplished by blasting vertical slices into an expansion slot having the height and width of the proposed stope. Stopes are limited by a crown pillar protecting the level above, rib pillars, and a sill pillar through which the ore collection system is cut. Pillars can be removed through large scale blasting, and 100 percent ore recovery can be achieved under perfect conditions. Figure 3.3.3. gives an schematic view of the sublevel stoping method.

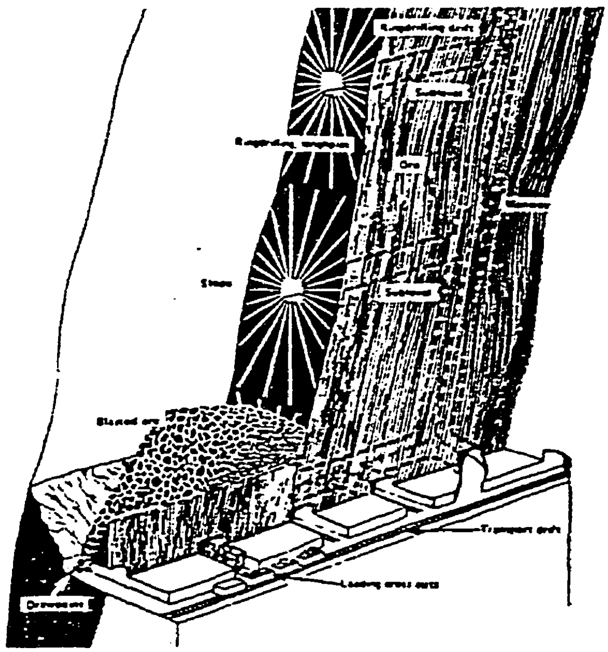

Figure 3.3.3. - Schematic View of Sublevel Stoping Method (Hamrin,1982)

Longhole equipment is used to perform production drilling. The utilization of large-diameter DTH (down-the-hole) drills are revolutionizing the method because of their directional accuracy (Pandey, 1984). The size of the highly mechanized drilling equipment limits the minimum width that can be mined, while high development costs require that a high production rate be maintained. However, efficient use of large-scale blasting makes sublevel stoping one of the lowest cost underground mining methods. Sublevel stoping also finds application for pillar recovery in cut and fill and other types of mining methods (Irvine, 1982; Bharti, Lebl, and Cornett, 1983).

In terms of explosive type, ANFO, the least expensive blasting agent, is commonly used. ANFO can be freely poured in down-holes or pneumatically loaded in up-holes. The timing sequence is critical to the success of the blast and the adequacy of fragmentation. Proper blasts also minimize the effect of excessive blast vibration and concussion effects underground.

3.3.4. Mining Costs

Room and Pillar

Based on a mine cost survey carried out by Mutmansky et al. (1992), the direct mining cost for an industrial mineral room and pillar mine is on average U.S.$ 2.62/ton (mid-1989). This low cost can be attributed to the nature of the industrial mineral deposits that are mined with conventional equipment. Table 3.3.1. lists the industrial mineral room and pillar mines with their mining cost breakdown.

Table 3.3.1. - Room and Pillar Industrial Mineral Mine Cost Data (Mid-1989 $)

(Modified from Mutmansky et al. (1992))

Mine	Size Million tons/yr	Cost Estimates (US$/ton moved)						
		Drilling Cost	Blasting Cost	Loading Cost	Haulage Cost	Hoisting Cost	Auxiliary Cost *	Direct Mining **
E	>5	0.12	0.29	0.18	0.43	0.43	1.16	2.61
F	1 to 5	0.18	0.43	0.55	0.23	0.43	0.80	2.62
Average		0.15	0.36	0.37	0.33	0.43	0.98	2.62
% of Direct Mining		6	14	14	13	16	37	

* Auxiliary costs include support, ventilation, pumping, etc.

** Sum of drilling, loading, haulage, hoisting and auxiliary costs

Auxiliary cost represents 37% of the total direct mining cost. Hoisting, loading, blasting and haulage have basically the same share of the direct mining cost with 16, 14, 14 and 13%, respectively. Drilling represents only 6% of the direct mining cost.

Sublevel Stoping

Sublevel stoping is a low-cost, high-production method, often selected for primary underground extraction when open pit mining is no longer economical (Hedberg, 1981). The use of large capacity equipment and mechanization enables the reduction of costs.

Figure 3.3.4. shows a breakdown of mining costs for a typical operation. The development intensive nature of the method is evidenced by the fact that development accounts for one third of total mining costs (Lawrence, 1982). Labour typically averages 40 to 50% of total stoping costs (Matikainen, 1981).

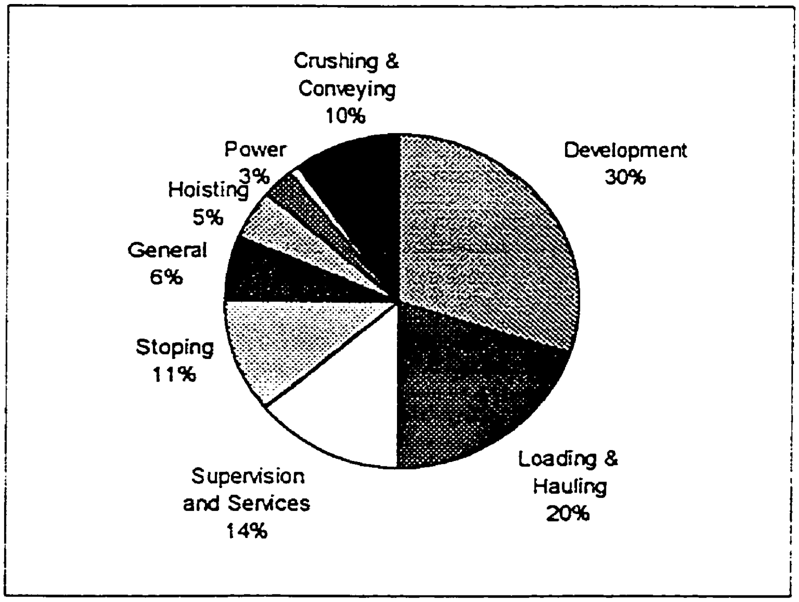

Figure 3.3.4. - Cost Break-down for Typical Sublevel Stope (Lawrence, 1982)

The Baltar mine, a limestone sublevel stoping operation in Brazil, has reported a direct mining cost of U.S.$ 2.59 per tonne (Benedetti et al., 1987). The mine has an annual capacity of 3 million tonnes with two 6-hour shifts. The entire mine output is used for cement production.

3.3.5. Environmental Issues

Although less conspicuous environmentally, underground mining still presents some environmental risks. According to Haycocks et al. (1992), ventilation fan noise and sinkholes are the only risks associated with underground limestone and dolomite operations. In terms of perceived magnitudes of environmental problems, sinkholes are of much higher concern than fan noise. Apart from these unique underground risks, there are some more general risks, such as water discharge, and water table and habitat disruption.

Permitting and zoning for underground mining are usually easier to obtain than for open pit mining. Zoning boards tend to view underground operations favourably as an alternative to surface mines.

Underground limestone mines have some unique characteristics that render them attractive for alternative uses after mining has been completed or when mined-out areas become available. The most important characteristics are constant temperature throughout the year and locations near or under heavily populated areas. Some of the uses for mined-out underground mines are: climate-controlled offices, warehouses, libraries, parking garages, waste disposal, mushroom culture. However, underground storage tanks could potentially contaminate underground water as they age and develop leaks. Because it is difficult to monitor the integrity of these underground tanks, their removal has been mandated by the federal government in the U.S.

3.4. Open Pit Conversion

There are numerous cases of mines which have started as open pit operations and were later converted into underground mines. For instance, the Kidd Creek Mine in Canada was initially exploited as an open pit until 1977, when it switched to the sublevel stoping method. Particularly in the coal industry, many mines often switch to underground methods when stripping ratios become too high. In the limestone industry, such conversion, although not common, also happens. In Iowa, most underground limestone mines are developed from the floors of existing pits (McKay and Bounk, 1987). More recently, Lafarge Corp. has decided to switch to an underground operation in one of their mines in the U.S. In some cases, the conversion is anticipated at the project planning stage, but often the decision comes along with the depletion of the open pit reserves. This is the case of the Palabora Mine, a large copper open pit in the Republic of South Africa, which has recently announced its decision to move to an underground operation. Some literature dealing with the economics of open pit conversion are summarized in the following pages.

The economics involved in the conversion decision have been described in detail by Nilsson (1982, 1992, 1997), who argues that in order to maximize the present value of the project, companies that operate open pit mines have to consider the possibility of a future underground operation. If technical conditions for underground mining do exist, the open pit may proceed as long as the cost of the last ton does not exceed that incurred for its underground extraction. Therefore, if there is favourable conditions for a future underground operation, the optimal depth of the open pit may be affected by the economics of underground mining. Nilsson presents a comprehensive case study for a hypothetical iron ore deposit. Various factors influencing the economics of the decision such as slope angle, interest rate and mining costs are thoroughly discussed. The concept is to analyse combinations of technically feasible open pit and underground mining scenarios in terms of present value. The scenario with the highest value is chosen.

Nilsson (1992) describes two alternative ways of calculating the final pit depth with the underground option:

1. Calculate the profit and cost of the ore associated with different pit depths. The moment of conversion should be when the cost of deepening the pit exceeds the cost of underground mining. If the cost of underground mining is higher than the corresponding revenue, the underground operation is not profitable.

2. Calculate the net present value of the cash flow profile of the entire mine with open pit and underground operations, for different pit depths. The one that yields the highest value is the preferred alternative.

Although both methods lead to the same conclusion, the first method does not consider the overall profitability of the entire operation, thereby preventing comparisons with other capital requirements. To determine profitability, the net present value of the entire operation must be considered.

Nilsson (1997) re-addresses some issues on how future underground mining may affect the optimum depth of the open pit. He explains how issues such as discount rate and investments must be handled in the analysis. Nilsson argues that without a proper discount rate, a pit will probably end up too deep.

Hedberg (1981) discusses how large scale underground mining could be an alternative to open pit mining. Major disadvantages of underground mining and the difference in operating and investment costs are presented. A new, large scale underground mining method is schematically proposed.

Camus (1992) proposes a methodology to determine the ultimate pit depth considering the possibility of an underground conversion. The methodology is suitable to work with pit optimizing routines such as the Lerchs-Grossmann, floating cone, network flow and Lemieux moving cone algorithm. It consists of running an open pit optimizer which takes into account an alternate cost resulting from underground extraction. According to this methodology, the block value is the difference between the economic values of the block exploited by open pit versus underground methods. A case study is presented for a porphyry copper deposit in Chile.

Weaver (1995) discusses the right time for a limestone producer to convert to underground mining and the misconception that limestone operators cannot afford underground mining. The advantages of converting to underground mining are highlighted with a quick cost comparison between open pit and underground mining methods in the U.S.

ECONOMIC BREAK-EVEN MODEL

4.1. Introduction

A description of the most relevant concepts and calculations in the economic break-even model is provided in this chapter. This chapter does not intend to be a detailed user's manual. Rather, it should be considered a guide to help understand the algorithms used in the model. It is worth noting that an extensive review of the underlying concepts is not part of the scope of this chapter.

The program code is written in the C language (Borland © C++, version 4.02). A full program listing is provided in the Appendix. Although the program runs under Windows 3.1 (or above), it is not a full Windows application.

Briefly, the model calculates and compares long-term open pit and underground after-tax cost flows based on either forecasted cost data supplied by the user, or on cost estimates derived from the O'Hara and Suboleski (1992) relationships. Revenues are ignored because they are common to both alternatives. The comparison of cost flows is based on the present worth concept of the remaining operational time span. The operational time span is defined as the time over which both open pit and underground operations are technically feasible, i.e. the total period under evaluation less time allowances for underground development, and a minimum underground life set by the user. The model compares the present worths of both alternatives, verifying whether the underground option becomes more economic in the long-term. A message with the recommended conversion time is displayed when this occurs. Figure 4.1.1. presents a flow chart of the model.

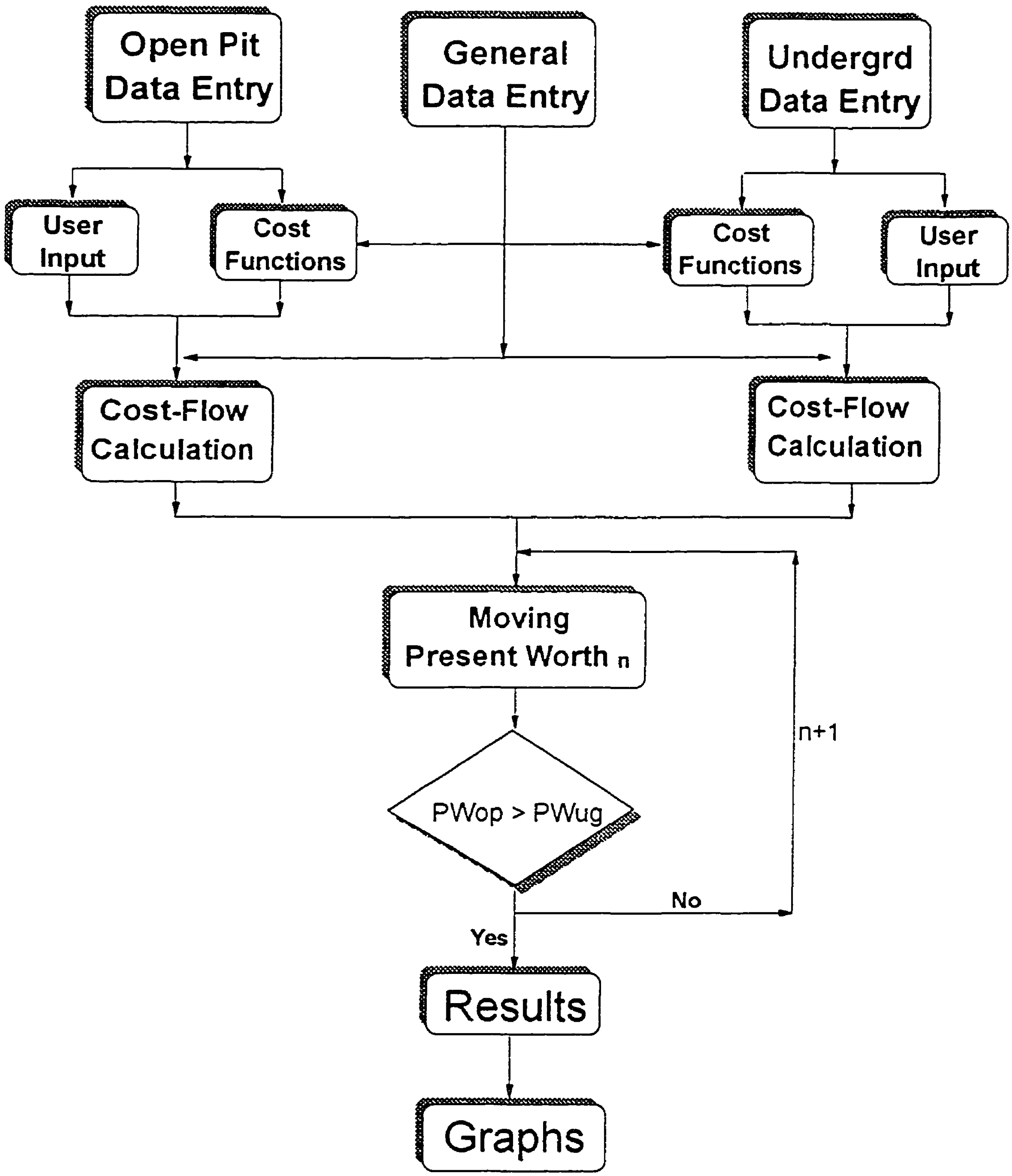

Figure 4.1.1. - Flow Chart of Break-Even Cost Model

This chapter is structured in three main sections. The first one deals with general information such as tax rate, inflation, discount rate and depreciation. A description of the open pit and underground cost estimation methods is presented in the second section. Finally, cost-flow and present worth calculations are detailed in the third section.

4.2. General Information

The following assumptions are built in the model:

End-of-Year Convention

In order to facilitate the economic analysis, all inflows and outflows incurred over a particular year are assumed to occur at the end of that year.

Tax Rate

A single tax rate is assumed in the program. This overall tax rate represents a combination of all taxes levied (federal, state, local) on the basis of a profit tax system. The mining operation is considered to be part of an integrated business. Therefore, the tax rate used is the one applicable to the enterprise as a whole.

The tax rate is used to calculate tax credits derived from the expensing of operating costs and the deduction of depreciation allowances.

Inflation Rate

There is no special provision for inflation in the model. All values are assumed to be in constant U.S. dollars.

Discount Rate

The discount rate accounts for the cost of capital. Such a discount rate is used to determine the present worth of the cost flows. A discrete discounting approach is used.

Unit System

The model has been developed for a limestone mine located in the United States. Thus, the Imperial units are used in the computations.

Depreciation

Depreciation is an allowable deduction which represents the exhaustion, wear, and tear of property used in a business, or of property held for the production of income. The purpose of depreciation is to provide a means by which a business can recover the capital required to replace its deteriorating capital assets.

Several methods for determining depreciation allowances exist. The most commonly used methods are:

- Straight-Line;
- Declining-Balance;
- Sum-of-the-Years'-Digits; and
- Unit of Production.

Although the total amount of depreciation deducted over the useful life of an asset is the same with all depreciation methods, a larger deduction in earlier years and less in later years is possible with accelerated depreciation methods such as the declining-balance method. The declining-balance method is used in this analysis. Equations 4.2.1. to 4.2.3. apply to the declining-balance depreciation method.

$$DA_j = IC\left[(1-r)^{j-1}\right]r \qquad (4.2.1.)$$

$$BV_j = IC\left[(1-r)^j\right] \qquad (4.2.2.)$$

$$ADA_j = IC\left[1-(1-r)^j\right] \qquad (4.2.3.)$$

where,

IC	=	Initial cost of asset
r	=	Depreciation rate (decimal)
DA_j	=	Depreciation allowance in year j
ADA_j	=	Accumulated depreciation allowance from year 1 to year j inclusively
BV_j	=	Book value of asset at the end of year j

4.3. Cost Estimation

Cost estimation is an essential part of the breakeven model since all calculations are based on cost analysis. Two procedures are available in the program to enter the cost information: user cost input, and cost functions. It is strongly recommended that, whatever option is used, the user adopts the same procedure for both open pit and underground options in order to maintain some coherence in the approach.
An explanation of both procedures follows.

4.3.1. User Cost Input

In this procedure, the planner should have all the long-term capital and operating cost estimates for either open pit and underground options calculated before-hand. Either preliminary or detailed cost estimates can be used as long as both mining options have the same degree of accuracy.

4.3.1.1. Open Pit Mine

Because an open pit operation is already in place, some of the cost information regarding labour, equipment maintenance and environment control are readily available. However, the long-term cost information must be based on a long-term mine plan for a time span of no less than 20 years. Such a timeframe is necessary to allow a direct comparison with the underground option. The mine plan primarily provides information on how the stripping ratio, haulage distance and environment will be affected over time. With such information, proper capital and operating costs estimates can be determined.

In the case of a greenfield mining project, capital costs consists of two components: fixed capital expenditures and working capital. The fixed capital costs refer to the total amount of money necessary to prepare the site, purchase primary and ancillary equipment and facilities, and cover other expenses associated with project start-up. However, in the present situation in which an open pit mine is in operation, the long-term capital costs relate to the purchase of new equipment due to either the replacement of existing units or the need of additional units in order to meet a higher demand. Drills, loaders and off-road trucks are usually the type of equipment to be considered. In some cases, however, the crushing system may need to be replaced or at least revamped during the period. This investment must also to be considered. Equipment conditions must be assessed individually and the equipment's economic life taken into account when preparing a schedule of acquisitions. Annual capital expenditures can then be determined based on either manufacturers' quotations or data from technical publications. The working capital represents the amount of money needed to begin and sustain the operation for a period of 1-3 months. In the present situation, working capital is already in the system and only additional working capital due to an increase in operating costs may be necessary.

Operating costs are defined as those ongoing costs incurred during normal operation of the project. In general, operating expenses can be divided into direct, indirect, and overhead costs. Direct costs primarily consist of labor and material charges. Indirect, or fixed costs, are expenditures independent of throughput such as administrative and safety labour, public relations and reclamation. Overhead expenses typically represent off-site charges for items such as marketing and sales, corporate officers, legal staff, and R&D. Because overhead costs do not depend on the extraction method, they do not need to be considered in the present case.

Direct long-term operating costs are affected by the long-term mine plan if stripping ratios and haulage distances increase over time. In this case, operating costs can be estimated by preparing budgets based on future equipment, personnel and supply requirements. Current cost levels can be used as a basis for estimating future costs. Indirect operating costs are likely to remain at current levels.

Environmental operating costs are not included in total operating costs and must be specified separately. All operating costs are expressed in terms of cost per ton of material (ore or waste).

4.3.1.2. Underground Mine

Underground cost estimates cannot rely on the current open pit cost information. A different approach must be used.

Pre-production capital expenditures consist of all costs necessary to bring the mine into production. For an underground mine, this comprises the cost of a shaft and/or ramp, hoisting plant, underground equipment (for drilling, loading and haulage), ventilation, water supply, crushing, repair shop, compressed air and water, and working capital. There are a variety of techniques available for capital cost estimation with a wide variation of degree of accuracy. Opinion estimating, the conference approach and the comparison approach are suitable for quickie estimates used mostly to screen unsound proposals without extensive engineering. For conceptual/order of magnitude estimates there are methods such as the unit cost (investment cost per unit of capacity) and capacity-cost curves. There are methods for preliminary (factored) estimates such as the cost ratio method and the component cost ratio method. Finally, for more accurate estimation, one should carry out a detailed cost estimate based on engineering drawings, layouts, equipment lists and so on. Contigency allowances and engineering fees must be factored into the pre-production capital expenditures. Obviously, the cost associated with each estimation method is exponentially proportional to the degree of accuracy required.

Pre-production capital expenditures are spread over the pre-production period according to some investment schedule.

There are fewer techniques utilized for developing operating cost estimates. Those used are essentially the same as those applied to capital cost estimates. Opinion estimating, the comparison approach, cost-capacity relationships and the component cost ratio method are some of the techniques used. Ultimately, accurate operating cost estimates must be developed from a detailed breakdown of major cost items. Estimates of powder factors, length drilled per bit, fuel consumption, and other variables must be made in order to develop a detailed operating cost estimate.

4.3.1.3. Sources of Cost Information

There are different sources of data related to operating costs. Information can be obtained internally within the organization and represents data collected from technical publications and other sources, or accumulated from similar projects. The major sources for this internal information are the Accounting Department, the Personnel Department, the Engineering and Geology Department, the Production Department, the U.S. Bureau of Labor Statistics and the Environmental Protection Agency. Unfortunately, a major source of information which has been discontinued is that of the U.S. Bureau of Mines, which provided a wide range of cost information. In Canada, the Canadian Mining Yearbook publishes annual surveys of the operating costs of established Canadian mines. Capital costs of major equipment components and construction costs can be derived from vendor quotations and manufacturer price lists, construction estimation handbooks and cost subscription services such as the Mining Cost Service published by Western Mine Engineering.

4.3.2. Cost Estimation Procedure

A brief review of available mining cost models is presented prior to describing the procedure for open pit and underground cost estimation implemented in this analysis.

4.3.2.1. Mining Cost Models

The development of general cost models requires considerable effort since the product must reflect a wide variety of mining aspects. All currently available models provide only preliminary cost estimates. Also, most models are restricted to certain mineral commodities and/or a specific region. For small precious metal mines in the western U.S., there is the Barrua/Bhappu (1987) model which should be used only for screening purposes due to its low degree of accuracy. The Pincock, Allen & Holt (PAH - 1988) model is also applicable to gold mines in the western U.S., but for larger scale operations. J.S. Redpath (1986) developed, under contract from the Canada Centre for Mineral and Energy Technology - CANMET, models to be applied for underground metal mines in Canada. The U.S. Bureau of Mines has also developed its Cost Estimating System (CES - 1984) which is presented in a manual of 2 volumes with more than 1200 pages. The CES consists of a series of costing components corresponding to specific mining/milling unit processes. For each component, cost curves and equations allow the generation of a cost for the unit process. The equations were derived using geometric regression techniques on actual and estimated costs of these unit processes. In 1991, the USBM's Western Field Operations Centre, located in Spokane, Washington, developed a set of mine and mill models designed for quick "pre-feasibility"cost estimation of mineral deposits in the desert region of the Southwest United States. The modelling approach developed, while based on the CES, was simplified and adapted accordingly. Regression analysis was used to generate capital and operating cost equations for a variety of open pit, underground and mill models. Total equations are broken down into 11 sub-equations, thus facilitating the application of inflation factors.

Another costing system was developed in Canada by A. Mular at the University of British Columbia. He first published a compilation of major mineral processing equipment capital costs in 1972. This was updated and expanded to include certain pieces of major mining equipment in 1977 and again in 1981, this time to include a section on preliminary capital and operating cost estimation based on O'Hara's cost estimation models.

One of the most complete factored-capacity cost estimating models for mining and processing projects was compiled and published by T.A. O'Hara (1980). Capital and operating costs for open pit mines, underground mines, and processing plants can be estimated from relationships based on relatively few variables. The estimating technique is based on cost data from not only Canadian but also foreign operations having comparable unit costs. The cost relationships derived are in terms of 1978 Canadian dollars.

The relationships were determined by fitting data to an equation having the general form given by Equation 4.3.2.1.

$$Q = K \bullet T^x \qquad (4.3.2.1.)$$

where Q is quantity or cost and T is a variable, such as production rate or size, causing changes in Q. The x value is determined to yield the lowest range of variation in K values across the widest range of T values for the data available.

One of the advantages of the relationships developed by O'Hara is that the overall project capital and operating costs can be made from a summation of cost items after judging the effect of specific local conditions on each component of the estimate. These specific conditions or factors can be assessed from knowledge of the local topography, climate and accessibility, and from drilling results and tests on core samples, and incorporated into the cost estimate with the use of the different relationships developed. As well, operating costs consist of the standard sub-components of labour, supplies, and administration and general services.

On contract from the Centre for Resource Studies at Queen's University (Ontario), O'Hara updated both his operating and capital cost equations to 1986 Canadian dollars and conditions. The revised equations account for technological changes in the quality of equipment, labour and supplies between 1978 and 1986, as well as real cost changes.

Most recently in 1992, O'Hara and Suboleski published a chapter on cost estimation in the SME Mining Engineering Handbook. Their cost relationships are based on actual costs of mine projects completed since 1980, which have been escalated by cost indices to the equivalent costs for the third quarter of 1988. The cost relationships implemented in this study are based on that publication and are detailed in the next section.

4.3.2.2. Open Pit Cost Estimation System

Annual capital expenditures are estimated based on information provided by the user regarding the current mining equipment (drills, loaders and trucks) and the long-term mine planning exercise. The mine plan produces information related to the daily amount of material (ore and waste) to be extracted which, through the use of O'Hara's equations, enables the prediction of the future mining equipment requirements. Here it is assumed that future mining equipment will be of the same type and size as the existing equipment. Future equipment requirements comprise replacements and additional equipment. The schedule for new acquisitions is worked out by offsetting, on an annual basis, the mining fleet available in the previous year from the requirements of that year. The schedule for replacement acquisitions takes into account the remaining life of the current equipment as well as the economic life of the acquired equipment. From the equipment acquisition schedule, and using O'Hara's equipment cost equations, the annual capital expenditures are determined. The O'Hara assumptions/equations used for the open pit capital expenditure estimates are listed below.

Open Pit Capital Expenditures

Drills

For tonnages up to 25 000 tpd, two drills of appropriate hole diameter should be used. Three drills are adequate for up to 60 000 tpd, and four or more drills are required for daily tonnages over 60 000.

The cost of drilling equipment is related to the number of drills *Nd* and the hole diameter *d* (inches). The cost of drilling equipment is given by Equation 4.3.2.2.

$$\textit{Drilling equipment cost (\$)} = Nd\ \$20\,000 d^{1.8} \tag{4.3.2.2.}$$

The cost includes a 25% allowance for drilling and accessory equipment.

Loaders

The number of loaders *Nl* with dipper size *S* (cubic yards) that are required to load a total of *Tp* tons of ore and waste daily is given by Equation 4.3.2.3.

$$Nl = 0.011\ (Tp)^{0.8} / S \tag{4.3.2.3.}$$

Nl is rounded down to the nearest integer.

The cost of loading equipment depends primarily on the number of loaders Nl and their bucket size S. The loading equipment cost is given by Equation 4.3.2.4.

$$\textit{Loading equipment cost (\$)} = Nl\ 510\,000 S^{0.8} \tag{4.3.2.4.}$$

Trucks

The total number of trucks *Nt* of capacity *t* (tons) required for the open pit fleet, plus an allowance for trucks under repair, is given by Equation 4.3.2.5.

$$Nt = 0.25\ Tp^{0.8} / t \tag{4.3.2.5.}$$

The cost of haulage equipment is given by Equation 4.3.2.6.

$$\textit{Haulage equipment cost (\$)} = Nt\ 20\ 400t^{0.9} \tag{4.3.2.6.}$$

Open Pit Operating Costs

It is assumed that the cost of mining ore is the same as that of mining waste. The cost of mining comprises drilling, blasting, loading and hauling. In addition, the cost of crushing the ore must be added to make this option comparable to the underground option in which the ore is crushed before hoisting it to surface. The cost of open pit mining can be assessed using the total ore and waste (*Tp*) mined daily and Equations 4.3.2.7. to 4.3.2.11.

$$\textit{Drilling cost per day (\$)} = 1.90\ Tp^{0.7} \tag{4.3.2.7.}$$

$$\textit{Blasting cost per day (\$)} = 3.17\ Tp^{0.7} \tag{4.3.2.8.}$$

$$\textit{Loading cost per day (\$)} = 2.67\ Tp^{0.7} \tag{4.3.2.9.}$$

$$\textit{Haulage cost per day (\$)} = 18.07\ Tp^{0.6} \tag{4.3.2.10.}$$

$$\textit{Crushing cost per day (\$)} = 7.90\ T_{ore}^{0.6} \tag{4.3.2.11.}$$

4.3.2.3. Underground Cost Estimation System

Basic technical information regarding the underground mine is required by the program. Prior to its use, it is essential that the user define variables such as extraction method, location of the orebody, means of access, presence of underground water and so on. Based on this information and the O'Hara relationships, the capital and operating costs for the underground operation can be determined. The equations used in this study are presented below.

Shaft Sinking

The cost of shaft sinking is a function of the cross-sectional area and depth of the shaft, which must be specified by the user. Such cost includes the fixed cost of erecting a temporary sinking plant and concreting the shaft collar. For shafts sunk to depths of 1000 ft or more, the major cost is the unit cost of shaft sinking. This unit cost tends to increase as the shaft deepens because of the longer time required for hoisting excavated muck. A circular concrete shaft is assumed. Fixed and variable costs are given by Equations 4.3.2.12. and 4.3.2.13., respectively.

$$\textit{Fixed costs for circular shafts (\$)} = 135\,000\ d^{0.5} \qquad (4.3.2.12.)$$

$$\textit{Variable costs for circular shafts (\$)} = 307\ d^{0.7} D_s^{1.05} \qquad (4.3.2.13.)$$

where d is the shaft diameter in ft and Ds is shaft depth in ft.

Hoisting Plant

The cost of the hoisting plant is a function of the size and type of hoist, hoisting rope speed, shaft depth, and the amount of ore to be hoisted per day. A double-drum hoist is assumed with a drum diameter D determined from Equation 4.3.2.15. and motor horsepower Hp determined from Equation 4.3.2.16. The rope speed S is given by Equation 4.3.2.14.

$$S = 1.6\ h^{0.5} T^{0.4} \text{ for hoisting } T \text{ tpd from a depth of } h \text{ ft} \qquad (4.3.2.14.)$$

$$D = 4.13\ T^{0.3} h^{0.14} \qquad (4.3.2.15.)$$

$$Hp = 0.5(D/100)^{2.4} S \tag{4.3.2.16.}$$

where S is in feet per minute and D is in inches.

The area A of the hoist room required for a double-drum hoist with a drum diameter of D can be assessed by Equation 4.3.2.17.

$$A = 0.10\ D^{2.2} \text{ (for one double-drum hoist)} \tag{4.3.2.17.}$$

where A is in square feet.

The cost of the hoist, hoist installation and hoistroom construction is given by Equations 4.3.2.18., 4.3.2.19., and 4.3.2.20., respectively.

$$\mathit{Cost\ of\ hoist\ (\$)} = 700\ (0.9\ D)^{1.4} Hp^{0.2} \tag{4.3.2.18.}$$

$$\mathit{Hoist\ installation\ (\$)} = 64\ D^{1.8} \tag{4.3.2.19.}$$

$$\mathit{Hoistroom\ construction\ (\$)} = 4.90\ A^{1.4} \tag{4.3.2.20.}$$

Headframe

The headframe cost will depend on the weight of steel required, which in turn depends on the height H of the headframe and the breaking strength of the hoist rope. The height of the headframe centre above the shaft collar is given by Equation 4.3.2.21. The weight of structural steel W is given by Equation 4.3.2.22. The cost of the headframe and ore bins are given by Equations 4.3.2.23. and 4.3.2.24., respectively.

$$H = 8.0\ T^{0.3} + 1.2\ S^{0.5} \tag{4.3.2.21.}$$

$$W = 0.12\ H^3 (D/100)^2 \tag{4.3.2.22.}$$

where H is in feet and W is in pounds.

$$\textit{Cost of headframe (\$)} = 19\ W^{0.9}$$ (for single-hoist headframe structure including shaft collar and foundations) (4.3.2.23.)

$$\textit{Cost of ore bins and skips (\$)} = 700\ T^{0.7} \tag{4.3.2.24.}$$

Mine Development

In general, the cost per foot for any excavation of cross-sectional area A is proportional to $A^{0.6}$. Thus one can relate the cost per unit length of any size excavation of cross-sectional area A to the unit cost of a standard 8x8-ft drift by the ratios given in Equations 4.3.2.25., 4.3.2.26. and 4.3.2.27. Lengths and cross-sectional areas for the excavations must be specified by the user.

Drifts or crosscuts of cross-sectional area A: $0.0825\ A^{0.6}$ (4.3.2.25.)

Ramps of cross-sectional area A: $0.0970\ A^{0.6}$ (4.3.2.26.)

Service Excavations of cross-sectional area A: $0.0948\ A^{0.6}$ (4.3.2.27.)

The cost ratios convert mine development openings into equivalent feet of 8x8-ft drift which has a cost of \$148/ft (1988 \$).

Drilling, Loading, and Haulage Equipment

All equipment for drilling, loading, and hauling ore, where such equipment is not fixed in place nor installed on foundations, is included in this section. It is assumed that the cost of equipment depends on the daily tonnage T and the width (i.e. thickness) W of ore. Equipment cost is derived from Equation 4.3.2.28.

$$\textit{Cost of equipment (S)} = 24\ 600\, T^{0.8} / W^{0.3} \tag{4.3.2.28.}$$

Mine Ventilation

The cost of the ventilation system can be estimated from the total installed horsepower (Hp) of all ventilation fans in the system. The total installed horsepower is a function of the volume of air and the average fan pressure required to move this quantity of air. The total installed horsepower is given by Equation 4.3.2.29. The cost of the ventilation system for a nonmetallic mine is given by Equation 4.3.2.30.

$$\textit{Total installed fan Hp} = 0.88\ T^{0.9} \tag{4.3.2.29.}$$

$$\textit{Cost of ventilation system (S)} = 7\ 500\ Hp^{0.6} \tag{4.3.2.30.}$$

Mine Pumping System

The cost of the drainage system is a function of the total installed horsepower of the pumps. The pump horsepower for dry mines with little inflow and a mine depth less than 1000 ft is provided by Equation 4.3.2.31. The total cost of the pumping system is given by Equation 4.3.2.32.

$$\textit{Installed horsepower Hp} = 8.0\ T^{0.5} \tag{4.3.2.31.}$$

$$\textit{Pumping system cost (\$)} = 1\,400\ Hp^{0.7} \tag{4.3.2.32.}$$

Water System

The water supply infrastructure cost depends on the amount of drilling and the type of drills. Equation 4.3.2.33. gives the cost of water supply infrastructure.

$$\textit{Cost of water supply (\$)} = 5\,300\ T^{0.4} \tag{4.3.2.33.}$$

Primary Crusher

Usually, the primary crusher is installed underground. This reduces problems with hangups in skip loading pockets, skip dumps, and conveyor transport. The cost of a jaw crusher and its installation is given by Equations 4.3.2.34. and 4.3.2.35., respectively.

$$\textit{Cost of jaw crusher (\$)} = 1\,370\ T^{0.6} \tag{4.3.2.34.}$$

$$\textit{Installation cost (\$)} = 210\ T^{0.7} \tag{4.3.2.35.}$$

Underground Maintenance Shop

The underground maintenance shop is normally located close to the shaft and its cost can be estimated by Equation 4.3.2.36.

$$\textit{Cost of maintenance shop (\$)} = 14\,600\ T^{0.4} \tag{4.3.2.36.}$$

Mine Compressor Plant

The cost of the compressor plant depends on the required capacity of compressed air C which is determined by Equation 4.3.2.37. The cost of plant is estimated with Equation 4.3.2.38.

$$C = 230\ T^{0.5} \tag{4.3.2.37.}$$

$$\textit{Cost of compressor plant (\$)} = 920\ C^{0.7} \tag{4.3.2.38.}$$

where C is in cubic feet per minute.

Air and Water Distribution

The cost of air and water distribution depends on the total compressor capacity and on the length of lateral development as given by Equation 4.3.2.40. The length L is a function of daily mined tonnage T and stope width W as given by Equation 4.3.2.39.

$$\textit{Length of lateral development } L = 1\,276\, T^{0.6} / W^{0.4} \tag{4.3.2.39.}$$

$$\textit{Cost of air and water distribution (\$)} = 2.80\ L^{0.9} C^{0.3} \tag{4.3.2.40.}$$

where L is in feet.

Electrical Distribution

The cost of substations and power cables underground depends on the average peak load of the mine. For underground mining facilities, the load UGL can be determined from Equation 4.3.2.41. The cost of underground substations is given by Equation 4.3.2.42.

$$UGL = 24.75T^{0.5} \tag{4.3.2.41.}$$

$$Cost\ of\ underground\ substations\ (\$) = 1600\ (UGL)^{0.9} \tag{4.3.2.42.}$$

where UGL is in kilowatts.

Total Underground Capital Cost

The total underground capital cost is determined by summing all the cost items previously discussed. These capital costs are spread over the underground development period according to some investment schedule.

Underground Operating Costs

The two most common underground mining methods for limestone are room and pillar and sublevel stoping (See Chapter 3). The user must specify which method is to be applied so that the proper costing equations are used. Daily operating costs for the room and pillar and sublevel stoping methods are given by Equations 4.3.2.43. and 4.3.2.44., respectively. These costs take into account drilling, blasting, loading, haulage and ground support, but neither crushing nor hoisting, which are given by Equations 4.3.2.45. and 4.3.2.46., respectively.

$$Room\ and\ pillar\ costs\ per\ day\ (\$) = 85\ T^{0.6} \tag{4.3.2.43.}$$

$$Sublevel\ stoping\ costs\ per\ day\ (\$) = 160\ T^{0.6} \tag{4.3.2.44.}$$

$$Crushing\ costs\ per\ day\ (\$) = 2.00\ T^{0.8} \tag{4.3.2.45.}$$

$$Hoisting\ costs\ per\ day\ (\$) = 4.70\ T^{0.8} \tag{4.3.2.46.}$$

4.3.2.4. Cost Escalation

All cost estimates derived from the O'Hara relationships are based on monetary values of the third quarter of 1988. These costs must be adjusted for inflation. In order to perform the escalation of past values, a reliable cost index must be selected. A cost index is the price level at a given time and place for a specified commodity or service, a class of commodities, activities or services, or a composite mixture of commodities, activities, and services, compared to the price level that existed at a base or reference time and place. It should be noted that cost indices are very general and should be used with care.

Many different cost indices have been developed and are published for a variety of purposes. For engineering studies there are those compiled by Chemical Engineering, Engineering News-Record, and Marshall and Swift.

Specifically for the North American mining industry, there is the Marshall and Swift Mining and Milling Equipment Cost Index and the indices compiled by the U.S. Bureau of Labor Statistics, which can be accessed via the Internet (http://stats.bls.gov).

The cost adjustment is accomplished by multiplying the out-of-date cost by a ratio of appropriate indices. Equation 4.3.2.47. shows how the adjustment is made.

$$Cost_{now} = Cost_{then} \times \left({Index_{now}}/{Index_{then}} \right) \quad (4.3.2.47.)$$

4.4. Cost Flow and Present Worth Calculations

After having determined the annual capital expenditures and operating costs for the open pit and underground operations, the annual cost flow profiles are determined. The cost flow is determined by Equation 4.4.1.

$$CFi = OCi\ (1 - t) + CEi - (t \cdot DAi) \qquad (4.4.1.)$$

where:	CFi	=	Cost Flow in year i
	OCi	=	Operating Cost in year i
	t	=	Tax Rate
	CEi	=	Capital Expenditure in year i
	DAi	=	Depreciation Allowance in year i

Present worth calculations for open pit and underground mine options are carried out differently because the underground cost flow profile must be shortened in order to match the remaining life of the open pit alternative. For the open pit option, the present worth at any given time t is the summation of all the discounted cost-flows from year i to year n, the last year of the planning period. However, for the underground option, year i always represents the first year of underground production. Present worths for both options are determined at the start of underground production. Thus, the cost flows for the underground development period must be appreciated to time t, and only production cost flows up to year n-i are considered, since year n-i+1 is beyond the predetermined planning period. The comparison is constrained by a minimum underground mine life. Figure 4.4.1. illustrates the present worth calculations and comparison for the open pit and underground options.

The comparison between present value equivalents is performed by verifying whether, at any given time t, the underground present worth becomes lower than the open pit present worth. The program displays the time when this happens. Otherwise the program indicates that this never happens.

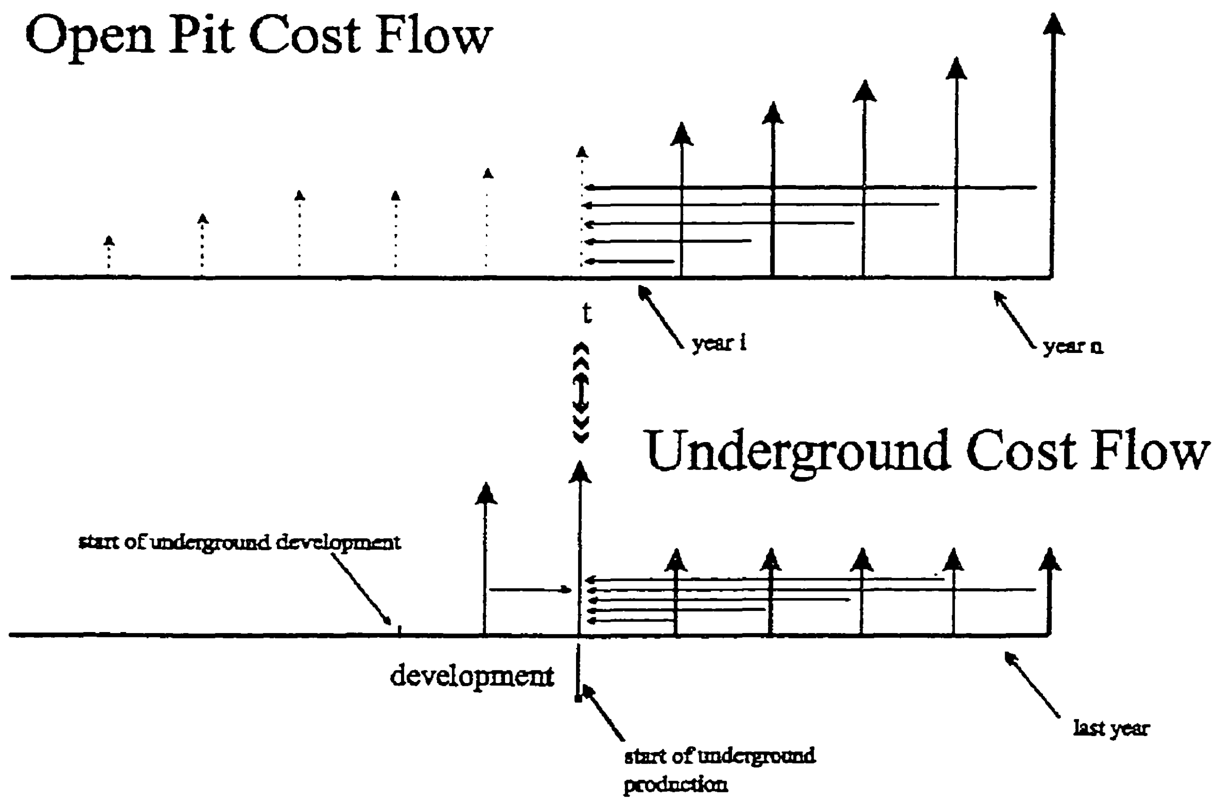

Figure 4.4.1. - Open Pit and Underground Present Worth Comparison

AS CEMENT INC. CASE STUDY

5.1. Introduction

In order to provide a better understanding of the problem under consideration, a case study is presented in this chapter. Although hypothetical, the data has been drawn from operating limestone mines around the world, offering a realistic example of the situation.

5.2. Case Study Description

AS Cement Inc. has been a medium-size cement producer in the Northeastern U.S. for the past 10 years. It owns a 3-kiln cement plant with an installed capacity of 2.0 million tons of clinker per year. Close by the plant is the limestone mine which supplies the basic raw material. Other raw ingredients for cement production, such as iron, silica and aluminum, are acquired from external sources. The company has no interest in the aggregate market since its limestone produces low quality stone.

The cement plant operates 7 days a week, 24 hours a day. The mine, however, has a 5 day a week (250 days per year), two 8-hour shift working schedule.

Limestone has been mined by the traditional blast-load open pit method since the start of the operation. The limestone currently being mined occurs in a flat lying bed of an average thickness of 150 ft and a width of 1500 ft. Average CaO and SiO2 content is 52 percent and 2 percent, respectively. Recently, however, the mine management team has been concerned about the following long term planning issues:

- increasing stripping ratios due to more overburden and waste material to remove;
- tighter quality requirement from the cement plant which requires less variation in MgO content;
- tougher national and local regulations for mine permitting, rendering the expansion of current operations more difficult.

Bearing this in mind, the management team decides to carry out a preliminary study of the technical alternatives available to minimize overall costs as well as supply limestone to the cement factory within a timeframe of 20 years. An external consultant is hired to provide some advice.

Based on his past experience, the consultant foresees the possibility of underground mining. There are already underground stone producers in the region and some previous deep drillholes on the property have indicated the presence of a flat lying bed of high quality limestone at a depth of 650 ft. The mine geologist estimates that this bed has an average thickness of 45 ft.

There are significant advantages to underground mining. For instance, overburden stripping, a costly operation in surface mining, is eliminated. Also, underground mining is less conspicuous environmentally and requires less land. Therefore, reclamation of disturbed land and its associated costs are reduced.

Although underground extraction is a technical alternative, the consultant is unsure of its economic worthiness when compared to a long-term open pit plan. Therefore, in order to determine the conversion time, i.e. the best time economically to switch from open pit to underground mining if indeed it exists, the consultant requests the following studies be carried out at the pre-feasibility level:

1. Long-term open pit mine plans for the next 5, 10 and 20 years, respectively Phase 1, Phase 2 and Phase 3. Stripping ratios, additional equipment requirements and average haulage distances are requested.

2. Underground mine capital and operating cost estimates. The production rate is to be kept at its current level.

Based on the above information the consultant will be able to carry out the economic analysis and assess the best time to switch to underground mining.

5.3. Open Pit Mine

5.3.1. Long-Term Planning

The stripping ratio and environmental impact play a major role in the open pit long-term plan. Due to its proximity to an urban area, the environmental aspect is very sensitive and a good relationship with the local community is essential. The community is located to the northwest of the property. Southwards, the operation is limited by a river and by poor quality limestone. This leads the long-term planning northwards, where the limestone bed becomes entwined with dolomite, increasing the amount of waste to be removed. The overburden consists of layers of sandstone and dolomite which have to be blasted before removal. A thin layer of soil is present as well. Nonetheless, enough limestone exists on the property to guarantee a 20-year open pit mine life.

The following basic parameters are assumed for long-term mine planning:

- bench height: 30 ft;
- minimum berm width: 20 ft;
- overall angle: 44°;
- face angle: 70°;
- annual production: 2.0 million tons of limestone.

Figure 5.3.1.1. shows a tri-dimensional view of the proposed long-term sequence. The 5-year mine plan (Phase 1) targets the northwest part of the deposit where there is less waste to be removed and the limestone quality is within the cement plant's requirements. Subsequently, in the 10-year plan (Phase 2), mining will be directed eastwards, until it reaches the east boundary of the deposit . For the last 10 years of the 20-year plan (Phase 3), mining will proceed towards the north, where waste is thicker and the limestone quality is lower.

Table 5.3.1.1. summarizes the main results of the long-term mine plan. Based on this information, capital and operating costs requirements for the next 20 years can be determined.

Table 5.3.1.1. - Summary of the Long-Term Open Pit Mine Plan

Period (Years)	Current	1 - 5	6 - 10	11 - 20
Limestone (million tons per year)	2.00	2.00	2.00	2.00
Waste (million tons per year)	1.20	2.30	3.20	4.20
Stripping Ratio (W/O)	0.60	1.15	1.60	2.10
Average Haulage Distance (miles)	0.23	0.34	0.38	0.52

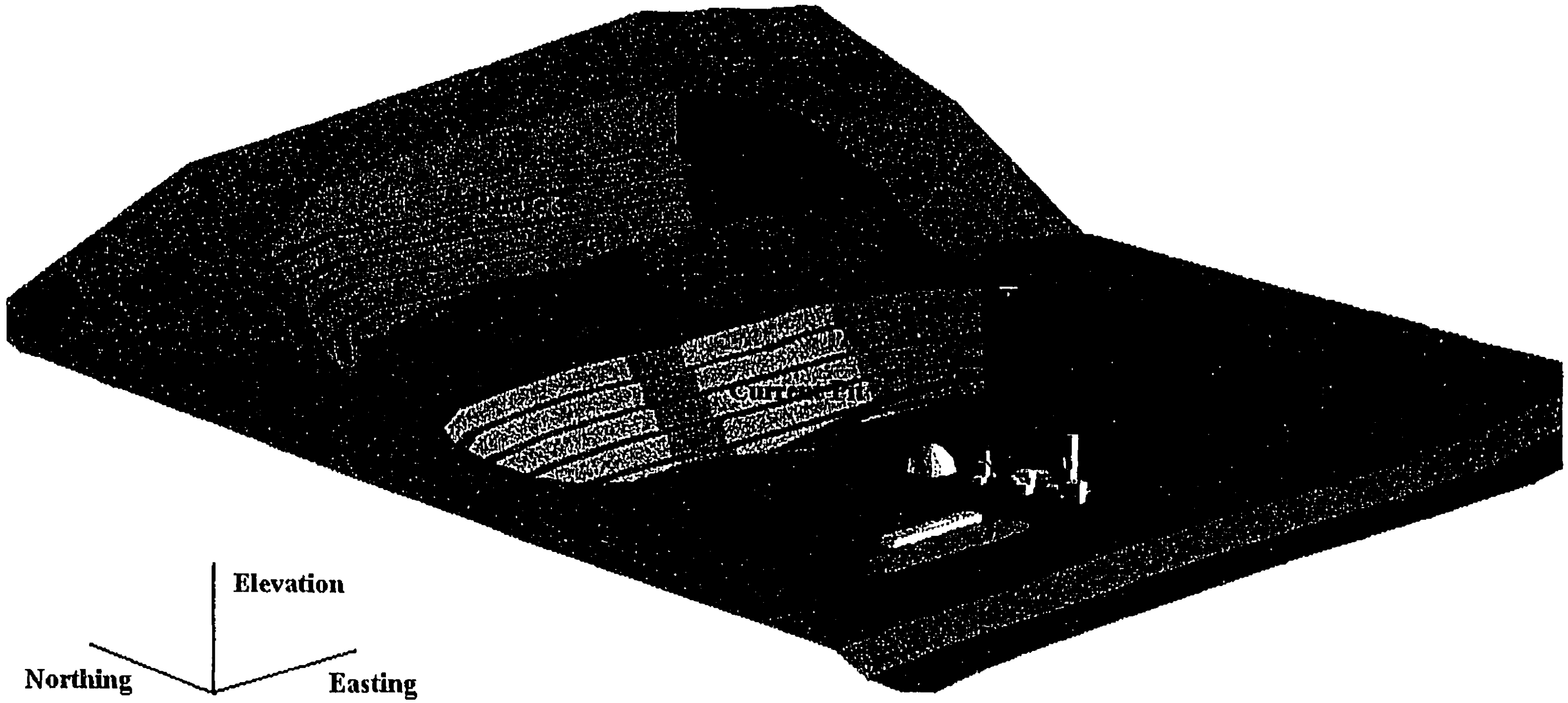

Figure 5.3.1.1. - Tri-dimensional View of the Long-Term Open Pit Mine Plan

5.3.2. Capital Cost - Mine Equipment

The mine currently operates with the equipment listed in Table 5.3.2.1. However, in the long-term, new equipment must be purchased to replace the existing equipment and also handle the higher stripping ratio levels. The schedule of equipment replacement is based on current operating hours in relation to the expected equipment life provided by the manufacturers. Table 5.3.2.2. details the status of existing equipment.

Table 5.3.2.1. - Current Open Pit Equipment

Equipment	Quantity
6.5" crawler rotary drill	2
ANFO loader truck	1
7 cu yd loader	3
55 ton mechanical drive haul truck	5
14' blade width crawler tractor	1

Table 5.3.2.2. - Existing Equipment Status

Equipment	Quantity	Current Hours	Hours per Year	Expected Life Hours	Expected Life Years	Expected Life Remaining Years
Drill A	1	12000	1500	20000	13	5
Drill B	1	7500	1500	20000	13	8
ANFO Truck A	1	6000	1500	20000	13	9
Loader A	1	12000	4000	30000	8	5
Loader B	1	15000	4000	30000	8	4
Loader C	1	20000	4000	30000	8	3
Haul Truck A	1	40000	4000	50000	13	3
Haul Truck B	1	40000	4000	50000	13	3
Haul Truck C	1	40000	4000	50000	13	3
Haul Truck D	1	30000	4000	50000	13	5
Haul Truck E	1	30000	4000	50000	13	5
Tractor A	1	5000	2500	12000	5	3

A description of the mine equipment acquisition schedule for the next 20 years is presented in Table 5.3.2.3. The item Replacement Units refers to future acquisitions to simply replace existing units. In order to handle the increasing waste material, additional units must be purchased - these are listed under Additional Units. Finally, Total Units sums both items for each year. The capital cost associated with each type of equipment is based on *Mine and Mill Equipment Costs - Western Mine Engineering (1995)*. The costs were escalated to 1996 dollars with the Marshall&Swift Equipment Index. The bottom line of Table 5.3.2.3. gives the total capital expenditure for each year of the 20-year period.

5.3.3. Operating Costs

The long-term operating costs are affected by the increasing stripping ratio, environmental restrictions and increasing haulage distance, in this sequence of importance. In order to have an estimate of the long-term operating costs, the long-term equipment schedule is used. It provides the amount of additional labour and equipment operating expenses for each year. Environmental costs encompass rehabilitation, community relations and environmental control.

Table 5.3.2.3. - Acquisition Schedule for Open Pit Equipment

Year / Equipment	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Drills (D)																				
Replacement Units						1			1											
Additional Units						1					1									
Total D Units						2			1		1									
Capital Cost ('000 US$)						895			447		447									
ANFO Trucks (AT)																				
Replacement Units										1										
Additional Units						1														
Total AT Units						1				1										
Capital Cost ('000 US$)						28				28										
Loaders (L)																				
Replacement Units				1	1	1						1	1	1						
Additional Units	1					1					1									
Replacement of Additional									1					1			1			
Total L Units	1			1	1	2			1		1	1	1	2			1			
Capital Cost ('000 US$)	564			564	564	1128			564		564	564	564	1128			564			
Haul Trucks (HT)																				
Replacement Units				3		2											3			
Additional Units	1					1					1									
Replacement of Additional														1						
Total HT Units	1			3		3					1			1			3			
Capital Cost ('000 US$)	488			1464		1464					488			488			1464			
Tractors (T)																				
Replacement Units				1					1						1					
Additional Units	1					1					1									
Replacement of Additional						1					2					3				
Total T Units	1			1		2			1		3				1	3				
Capital Cost ('000 US$)	309			309		619			309		928				309	928				
Total Capital Cost																				
D+AT+L+HT+T ('000 US$)	1361			2337	564	4134			1321	28	2428	564	564	1616	309	928	2028			

Current Operating Costs

The determination of the current operating costs is described below:

- Labour

The current budget by shift is provided in Table 5.3.3.1. Labour costs are based on \$15.50 per hour and a 56.5% payroll burden. Salary costs are based on an average of \$60 000 per year and a 28.2% burden. The mine operates on a 5-day-a-week (250 days per year) two 8-hour-shifts per day schedule.

Table 5.3.3.1. - Current Labour Budget

Labour Budget							
Shift Hourly	1 (Persons)	2 (Persons)	Total (Persons)	Base Rate $/hr	Burden %	Total Rate $/hr	Annual Payroll '000 US$
Loader Operator	3	3	6	15.5	56.5	24.26	291
Truck Operator	5	5	10	15.5	56.5	24.26	485
Tractor Operator	1	1	2	15.5	56.5	24.26	97
Driller	2	2	4	15.5	56.5	24.26	194
Blaster	1	1	2	15.5	56.5	24.26	97
Utility	2	2	4	15.5	56.5	24.26	194
Crusher Operator	1	1	2	15.5	56.5	24.26	97
Total, Hourly	15	15	30				1456
Salary				'000$/year			
Superintendent	0	1	1	60	28.2	77	77
Shift Foreman	1	1	2	60	28.2	77	154
Maintenance Chief	0	1	1	60	28.2	77	77
Total, Salary	1	3	4				308

- Equipment

Equipment hourly operating costs (excluding operator costs) are presented in Table 5.3.3.2. The costs are divided into parts (excluding tires), maintenance labour, diesel fuel, electric power, lube, and tires.

Table 5.3.3.2. - Current Equipment Operating Costs

Equipment Operating Costs (US$/hr)								
Type	Parts	Maint Labour	Diesel Fuel	Electric Power	Lube	Tires	Total per Unit	Total per Typ
Crusher	22.1	16.5	0.0	9.5	3.0	0.0	51.1	51.1
Loader	7.3	4.8	7.4	0.0	3.6	10.6	33.6	100.7
Haul Truck	4.4	3.8	8.0	0.0	3.4	7.4	27.0	134.9
Tractor	4.9	4.3	4.5	0.0	2.0	0.0	15.7	15.7
Drill	8.9	6.8	1.5	0.0	5.2	0.0	22.4	44.7
ANFO truck	0.6	0.4	3.5	0.0	0.4	0.4	5.3	5.3
Total							154.9	352.3

- Explosives

Annual expenses of ANFO and caps for the open pit operation total US$ 800 000.

- Environment

Costs associated with dumping site reclamation, community relations and environmental control total US$ 200 000 for the current year.

- Miscellaneous

The miscellaneous operating costs total US$ 400 000 per year. It comprises all the costs not directly related to labour, equipment, explosives, and environment control.

- Total Annual Operating Costs

Based on the cost components previously presented and the equipment working schedule, the annual operating costs total US$ 4 572 720. Hence, the operating cost per ton of limestone is US$ 2.29, and the cost per ton of material handled is US$ 1.43 when the stripping ratio is taken into account. A detailed break-down of the current operating costs is illustrated in Figure 5.3.3.1.

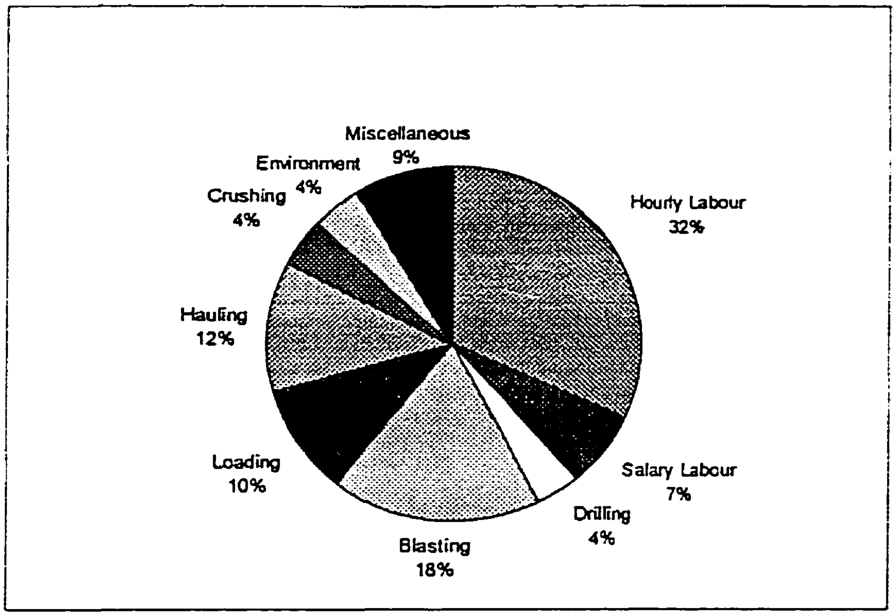

Figure 5.3.3.1. - Current Operating Cost Break-down

Long-Term Operating Costs

Phase 1 - Years 1 to 5

- Labour

Table 5.3.3.3. presents the annual hourly and salary labour budget for years 1 to 5.

Table 5.3.3.3. - Phase 1 Labour Budget

Labour Budget							
Shift Hourly	1 (Persons)	2 (Persons)	Total (Persons)	Base Rate $/hr	Burden %	Total Rate $/hr	Annual Payroll '000 US$
Loader Operator	4	4	8	15.5	56.5	24.26	388
Truck Operator	6	6	12	15.5	56.5	24.26	582
Tractor Operator	2	2	2	15.5	56.5	24.26	97
Driller	2	2	4	15.5	56.5	24.26	194
Blaster	1	1	2	15.5	56.5	24.26	97
Utility	2	2	4	15.5	56.5	24.26	194
Crusher Operator	1	1	2	15.5	56.5	24.26	97
Total, Hourly	18	18	36				1650
Salary				'000$/year			
Superintendent	0	1	1	60	28.2	77	77
Shift Foreman	1	1	2	60	28.2	77	154
Maintenance Chief	0	1	1	60	28.2	77	77
Total, Salary	1	3	4				308

- Equipment

Table 5.3.3.4. presents the equipment operating costs for years 1 to 5.

Table 5.3.3.4. - Phase 1 Equipment Operating Costs

Equipment Operating Costs (US$/hr)								
Type	Parts	Maint Labour	Diesel Fuel	Electric Power	Lube	Tires	Total per Unit	Total per Typ
Crusher	22.1	16.5	0.0	9.5	3.0	0.0	51.1	51.1
Loader	7.3	4.8	7.4	0.0	3.6	10.6	33.6	134.3
Haul Truck	4.4	3.8	8.0	0.0	3.4	7.4	27.0	161.8
Tractor	4.9	4.3	4.5	0.0	2.0	0.0	15.7	31.3
Drill	8.9	6.8	1.5	0.0	5.2	0.0	22.4	44.7
ANFO truck	0.6	0.4	3.5	0.0	0.4	0.4	5.3	5.3
Total							154.9	428.5

- Explosives

For years 1 to 5, annual expenses of ANFO and caps for the open pit operation are estimated at US$ 1 000 000.

- Environment

Costs associated with dumping site reclamation, community relations and environmental control are estimated at US$ 250 000 per annum for years 1 to 5.

- Miscellaneous

Miscellaneous operating costs are estimated at US$ 500 000 per annum for years 1 to 5.

- Total Annual Operating Costs

Based on the cost components previously presented and the equipment working schedule, the annual operating costs total US$ 5 421 560. Hence, the operating cost per ton of limestone is US$ 2.70, and the cost per ton of material handled is US$ 1.26 when the stripping ratio is taken into account. A detailed break-down of the operating cost estimates for Phase 1 is illustrated in Figure 5.3.3.2.

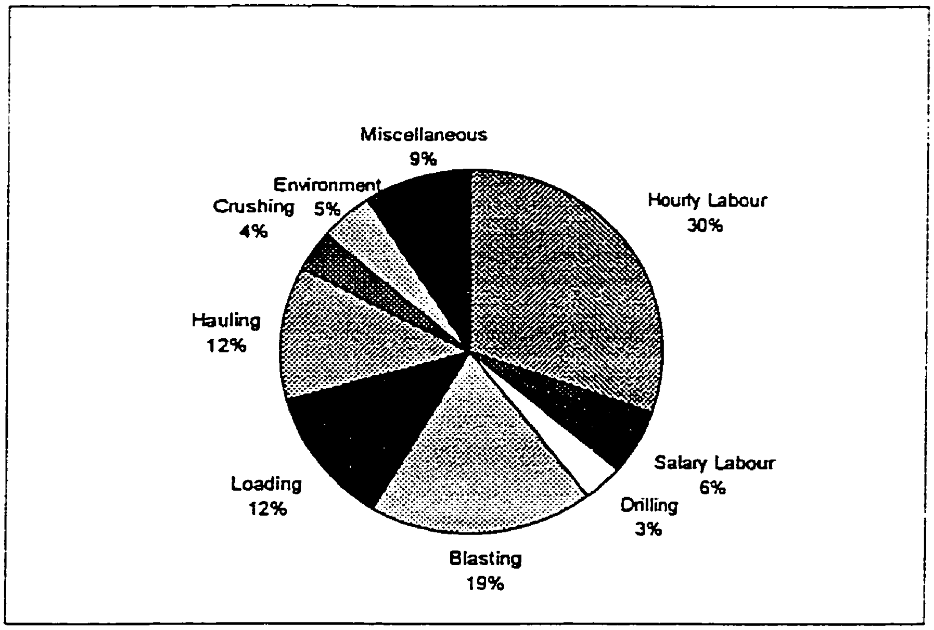

Figure 5.3.3.2. - Phase 1 Operating Cost Break-down

Phase 2 - Years 6 to 10

- Labour

Table 5.3.3.5. presents the annual hourly and salary labour budget for years 6 to 10.

Table 5.3.3.5. - Phase 2 Labour Budget

Labour Budget							
Shift Hourly	1 (Persons)	2 (Persons)	Total (Persons)	Base Rate $/hr	Burden %	Total Rate $/hr	Annual Payroll '000 US$
Loader Operator	5	5	10	15.5	56.5	24.26	485
Truck Operator	7	7	14	15.5	56.5	24.26	679
Tractor Operator	3	3	2	15.5	56.5	24.26	97
Driller	3	3	6	15.5	56.5	24.26	291
Blaster	1	1	2	15.5	56.5	24.26	97
Utility	3	3	6	15.5	56.5	24.26	291
Crusher Operator	1	1	2	15.5	56.5	24.26	97
Total, Hourly	23	23	46				2038
Salary				'000$/year			'000$/year
Superintendent	0	1	1	60	28.2	77	77
Shift Foreman	1	1	2	60	28.2	77	154
Maintenance Chief	0	1	1	60	28.2	77	77
Total, Salary	1	3	4				308

- Equipment

Table 5.3.3.6. presents the equipment operating costs for years 6 to 10.

Table 5.3.3.6. - Phase 2 Equipment Operating Costs

Equipment Operating Costs (US$/hr)								
Type	Parts	Maint Labour	Diesel Fuel	Electric Powe	Lube	Tires	Total per Unit	Total Per Typ
Crusher	22.1	16.5	0.0	9.5	3.0	0.0	51.1	51.1
Loader	7.3	4.8	7.4	0.0	3.6	10.6	33.6	167.9
Haul Truck	4.4	3.8	8.0	0.0	3.4	7.4	27.0	188.8
Tractor	4.9	4.3	4.5	0.0	2.0	0.0	15.7	47.0
Drill	8.9	6.8	1.5	0.0	5.2	0.0	22.4	67.1
ANFO truck	0.6	0.4	3.5	0.0	0.4	0.4	5.3	10.5
Total							154.9	532.29

- Explosives

For years 6 to 10, annual expenses of ANFO and caps for the open pit operation are estimated at US$ 1 200 000.

- Environment

Costs associated with dumping site reclamation, community relations and environmental control are estimated at US$ 300 000 per annum for years 6 to 10.

- Miscellaneous

Miscellaneous operating costs are estimated at US$ 600 000 per annum for years 6 to 10.

- Total Annual Operating Costs

Based on the cost components previously presented and the equipment working schedule, the annual operating costs total US$ 6 575 000. Hence, the operating cost per ton of limestone is US$ 3.29, and the cost per ton of material handled is US$ 1.26 when the stripping ratio is taken into account. A detailed break-down of the operating cost estimates for Phase 2 is illustrated in Figure 5.3.3.3.

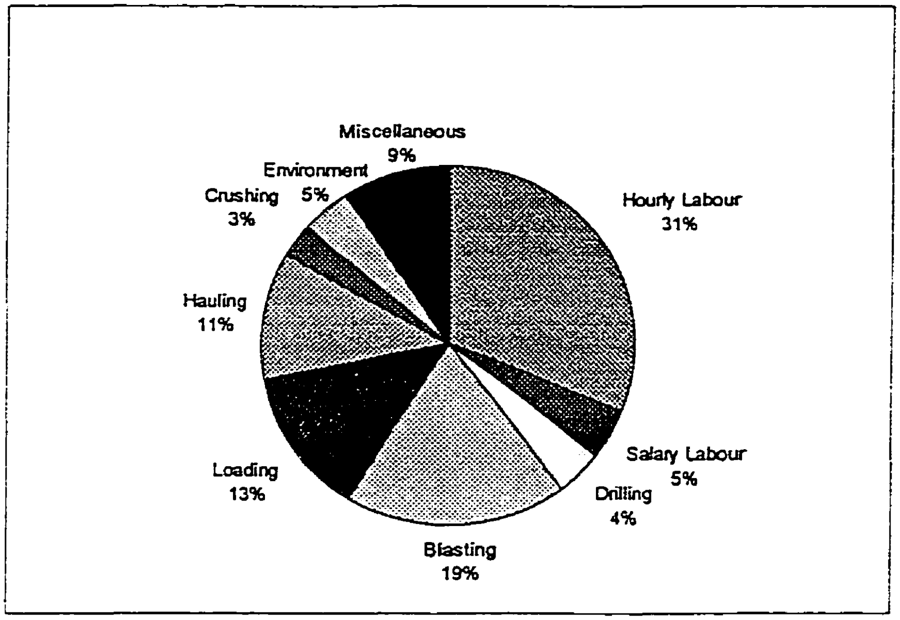

Figure 5.3.3.3. - Phase 2 Operating Cost Break-down

Phase 3 - Years 11 to 20

- Labour

Table 5.3.3.7. presents the annual hourly and salary labour budget for years 11 to 20.

Table 5.3.3.7. - Phase 3 Labour Budget

Labour Budget							
Shift Hourly	1 (Persons)	2 (Persons)	Total (Persons)	Base Rate $/hr	Burden %	Total Rate $/hr	Annual Payroll '000 US$
Loader Operator	6	6	12	15.5	56.5	24.26	582
Truck Operator	8	8	16	15.5	56.5	24.26	776
Tractor Operator	4	4	2	15.5	56.5	24.26	97
Driller	4	4	8	15.5	56.5	24.26	388
Blaster	2	2	4	15.5	56.5	24.26	194
Utility	4	4	8	15.5	56.5	24.26	388
Crusher Operator	1	1	2	15.5	56.5	24.26	97
Total, Hourly	29	29	58				2523
Salary				'000$/year			'000$/year
Superintendent	0	1	1	60	28.2	77	77
Shift Foreman	1	1	2	60	28.2	77	154
Maintenance Chief	0	1	1	60	28.2	77	77
Total, Salary	1	3	4				308

- Equipment

Table 5.3.3.8. presents the equipment operating costs for years 11 to 20.

Table 5.3.3.8. - Phase 3 Equipment Operating Costs

Equipment Operating Costs (US$/hr)								
Type	Parts	Maint Labour	Diesel Fuel	Electric Powe	Lube	Tires	Total per Unit	Total per Typ
Crusher	22.08	16.46	0	9.51	3.03	0	51.08	51.08
Loader	7.29	4.75	7.37	0	3.58	10.58	33.57	201.42
Haul Truck	4.37	3.79	8	0	3.44	7.37	26.97	215.76
Tractor	4.93	4.28	4.47	0	1.97	0	15.65	62.6
Drill	8.89	6.76	1.51	0	5.2	0	22.36	89.44
ANFO truck	0.57	0.43	3.54	0	0.35	0.38	5.27	10.54
Total							154.9	630.84

- Explosives

For years 11 to 20, annual expenses of ANFO and caps for the open pit operation are estimated at US$ 1 400 000.

- Environment

Costs associated with dumping site reclamation, community relations and environmental control are estimated at US$ 400 000 per annum for years 11 to 20.

• Miscellaneous

Miscellaneous operating costs are estimated at US$ 700 000 per annum for years 11 to 20.

• Total Annual Operating Costs

Based on the cost components previously presented and the equipment working schedule, the annual operating costs total US$ 7 854 400. Hence, the operating cost per ton of limestone is US$ 3.93, and the cost per ton of material handled is US$ 1.27 when the stripping ratio is taken into account. A detailed break-down of the operating cost estimates for Phase 3 is illustrated in Figure 5.3.3.4.

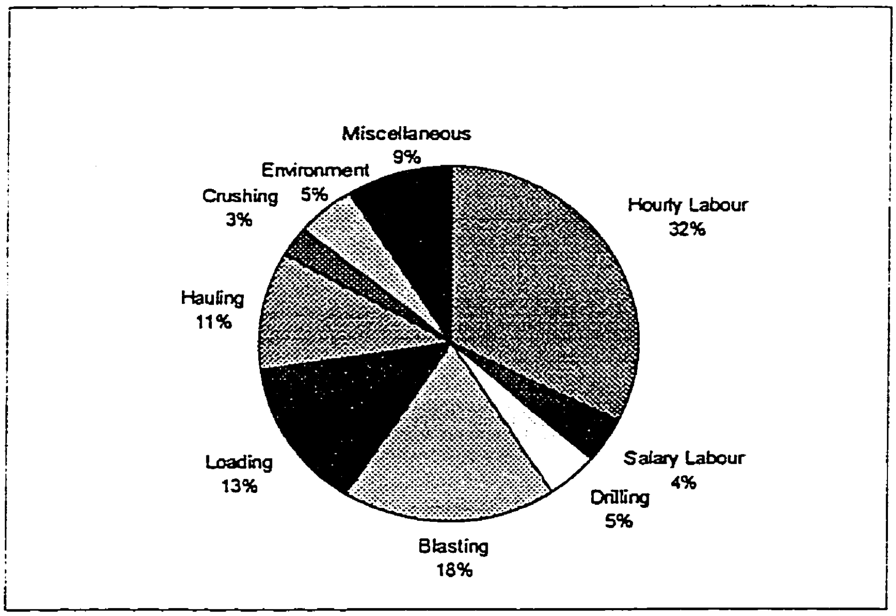

Figure 5.3.3.4. - Phase 3 Operating Cost Break-down

5.4. Underground Mine

The mining method suitable for the deeper flat-lying limestone bed is room and pillar with drilling and blasting excavation. The average thickness of 45 ft allows the use of a full face heading and benching operation, leaving a roof beam of 5 ft to ensure security in the working areas. Loaders and trucks will be used for extraction. According to production requirements and a preliminary rock mechanics study, a room width of 45 ft with pillars of 45 by 45 ft are required. The most likely production scenario, when the mine is in full production, would be 3 benches and 5 headings per day, totalling 8 000 tons daily. Figure 5.4.1. gives a schematic view of the planned room and pillar operation. The relatively impermeable nature of the overlying rock and the presence of clay beds should prevent any significant water infiltrations in the underground workings.

The mine accesses will consist of a 15 percent ramp with a 16 by 18 ft cross-section, and a 14 ft diameter shaft. The ramp will provide easy access to the underground workings, especially for the large mobile equipment that would otherwise have to be disassembled, lowered via the shaft, and reassembled. The ramp will also be equipped with a conveyor that will hoist the limestone to surface. A 15 percent ramp will have a length of 4 800 ft. The shaft is required for safety reasons and also to provide air to the underground workings. Preliminary shaft and ramp portal locations are shown in Figure 5.4.1.

The time required to convert the mine to an underground operation is estimated to be two years. Over this period, open pit production will continue to supply the raw material needed for the plant.

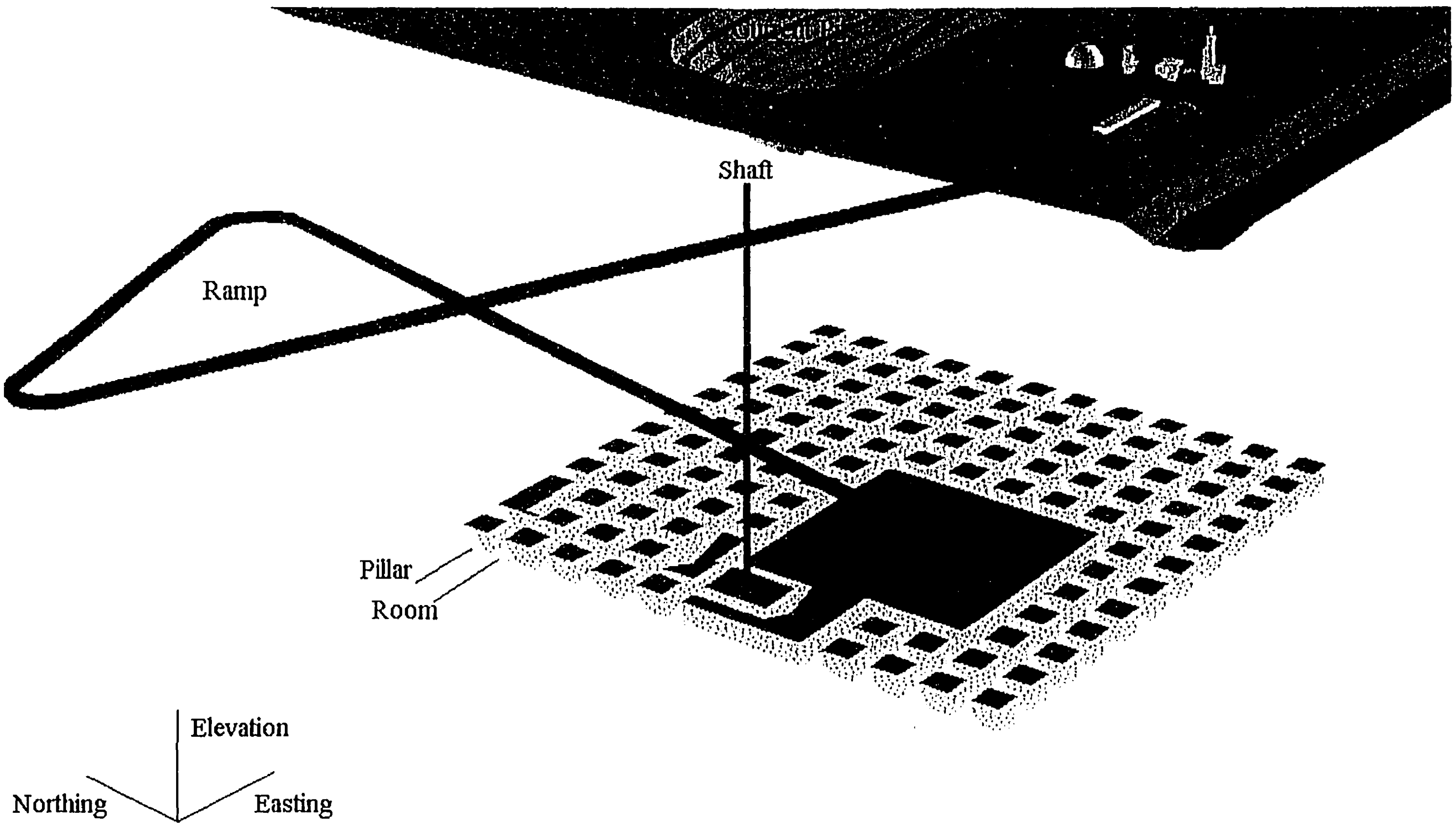

Figure 5.4.1. - Schematic View of the Underground Mine

5.4.1. Capital Cost Estimate

The capital cost estimate is divided into three major components: Mine Development, Hoisting and Mining Equipment. Adding 8% for preproduction engineering and management and an additional 10% contingency, the total estimated capital cost over the 2-year period is US$ 18.51 million. Table 5.4.1.1. summarizes the capital expenditure schedule.

Table 5.4.1.1. - Preproduction Capital Expenditure Schedule

Schedule of Capital Costs	millions US$		
Pre-production Year	1	2	Total
Ramp Development	3.31	1.66	4.97
Shaft Development	1.30	0.65	1.95
Hoisting		2.90	2.90
Underground Equipment		5.77	5.77
Subtotal	4.61	10.97	15.58
Preproduction Engineering, and Management (8%)	0.37	0.88	1.25
Subtotal	4.98	11.85	16.83
Contingency	0.50	1.18	1.68
TOTAL	5.48	13.03	18.51

Mine Development

Mine development consists of all excavations needed to provide access to and services for the mining activities. Two major items are required: a 15 percent ramp and a ventilation / emergency shaft. Both will be developed by contractors and the cost estimate is based on preliminary contacts.

Ramp Excavation

For ramp excavation, roadheader equipment will be used. It is estimated that a 15 month period will be required to complete the ramp with 2 crews working on a 2-shift-per-day and a 5-day-per-week schedule. A break-down of the costs involved is given in Table 5.4.1.2.

Table 5.4.1.2. - Ramp Excavation Cost Estimate

Ramp Excavation Based on 4 800 ft of 16ft wide x 18 ft high arched ramp	millions US$
Mobilization / demobilization	0.280
Plant set-up / teardown	0.175
Portal	0.700
Excavation - host rock	3.815
TOTAL	4.970

Shaft Excavation

Due to rock quality, the shaft must be sunk by drilling and blasting. Raise boring cannot be used. A total of 13 months is required to complete the work with 2 crews working 2 shifts per day. A break-down of the costs involved is given in Table 5.4.1.3.

Table 5.4.1.3. - Shaft Excavation Cost Estimate

Shaft Excavation Based on 650 ft of 14 ft diameter shaft	millions US$
Mobilization / demobilization	0.420
Plant set-up / teardown	0.280
Hoist set-up / teardown	0.525
Collar	0.105
Excavation - host rock	0.616
TOTAL	1.946

Hoisting

The crushed limestone will be hoisted to surface by the ramp conveyor system. An emergency hoist will be installed in the shaft for safety reasons. Table 5.4.1.4. lists the cost of the conveyor system and emergency hoist.

Table 5.4.1.4. - Hoisting Cost Estimate

Hoisting	millions US$
15% Slope Conveyor	2.7
Escape Hoist	0.2

Underground Equipment

The underground equipment list is given in Table 5.4.1.5. The list contains the major production and auxiliary equipment required for the operation. Cost estimates are based on *Mine and Mill Equipment Costs (1995).*

Table 5.4.1.5. - Underground Equipment Cost Estimate

Underground Equipment	millions US$
Mine Fan	0.30
8 yd Loaders (2 @ $400k)	0.80
40 ton Trucks (4 @ $315k)	1.26
Twin Boom Face Drill	0.55
Bench Drill	0.15
Scaler	0.28
High Reach Work Platform	0.15
ANFO Loader	0.20
Diesel Pickups (3 @ $20k)	0.06
Crushing System	1.05
Auxiliary Fans (4 @ $30k)	0.12
Electrical Distribution	0.50
Compressed Air/Water Distribution	0.10
Miscellaneous Equipment	0.25
TOTAL	5.77

The replacement schedule for the major items is in Table 5.4.1.5. and the total capital expenditure for each year is given in Table 5.4.1.6.

Table 5.4.1.6. - Replacement Schedule for Underground Equipment

Year Equipment	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Fan System (F)																				
Replacement Units															1					
Capital Cost ('000 US$)															420					
Drills (D)																				
Replacement Units													2							
Capital Cost ('000 US$)													700							
Loaders (L)																				
Replacement Units								2								2				
Capital Cost ('000 US$)								800								800				
Trucks (T)																				
Replacement Units													4							
Capital Cost ('000 US$)													1260							
ANFO Loader / Pickups (P)																				
Replacement Units								4												
Capital Cost ('000 US$)								260												
Total Capital Cost																				
F+D+T+L+P ('000 US$)								1060					1960		420	800				

5.4.2. Operating Cost Estimate

In order to determine the operating cost estimates, four major cost categories were defined, i.e. labour, equipment, explosives, and miscellaneous. A detail description of each category follows.

- Labour

Because of the harsh underground environment, the base hourly rate for underground workers is higher than for their surface counterparts. The underground hourly rate is US$ 18.00. Table 5.4.2.1. gives the annual hourly and salary labour estimates.

Table 5.4.2.1. - Underground Labour Budget

Labour Budget								
Shift Hourly	1 (Persons)	2 (Persons)	3 (Persons)	Total (Persons)	Base Rate $/hr	Burden %	Total Rate $/hr	Annual Payroll '000 US$
Loader Operator	1	2	2	5	18	56.5	28.2	282
Truck Operator	0	4	4	8	18	56.5	28.2	451
Scaler/Bolter	2	0	0	2	18	56.5	28.2	113
Driller	2	0	2	4	18	56.5	28.2	225
Blaster	2	0	0	2	18	56.5	28.2	113
Utility	1	1	1	3	18	56.5	28.2	169
Mechanic	2	2	2	6	18	56.5	28.2	338
Total, Hourly	10	9	11	30				1690
Salary					'000$/year			
Superintendent	1	0	0	1	60	28.2	77	77
Shift Foreman	0	1	1	2	60	28.2	77	154
Maintenance Chief	1	0	0	1	60	28 2	77	77
Total, Salary	2	1	1	4				308

- Equipment

Table 5.4.2.2. gives the equipment operating costs for the underground operation.

Table 5.4.2.2. - Underground Equipment Operating Costs

Equipment Operating Costs (US$/hr)								
Type	Parts	Maint.Labour	Diesel Fuel	Electric Power	Lube	Tires	Total per Unit	Total per Type
Crusher	22.08	16.46	0	9.51	3.03	0	51.08	51.08
Loader	7.29	4.75	7.37	0	3.58	10.58	33.57	67.14
Haul Truck	4.08	3.99	4.4	0	2.74	5.4	20.61	82.44
Twin Boom Face Drill	8.45	7.57	0	5.63	2.99	0.13	24.77	24.77
Bench Drill	8.89	6.76	1.51	0	5 2	0	22.36	22.36
ANFO Loader	3.63	2.65	1.65	0	1.12	0.25	9.3	9.3
Fan System	8.17	11.07			1.54		20.78	20.78
Scaler	5.54	4.05	1.65		1.68	0.25	13.15	13.15
Conveyor	16	7.5		27	5		55.5	55.5
Total							251.12	346.52

- Explosives

Annual expenses of ANFO and caps for the underground operation are estimated at US$ 650 000.

- Miscellaneous

Miscellaneous operating costs are estimated at US$ 600 000 per year.

- Total Annual Operating Costs

Based on the cost components previously presented and the equipment working schedule, the annual operating costs total US$ 5 149 220. Hence, the operating cost per ton of limestone is US$ 2.57. A detailed break-down of the operating cost estimate is illustrated in Figure 5.4.2.1.

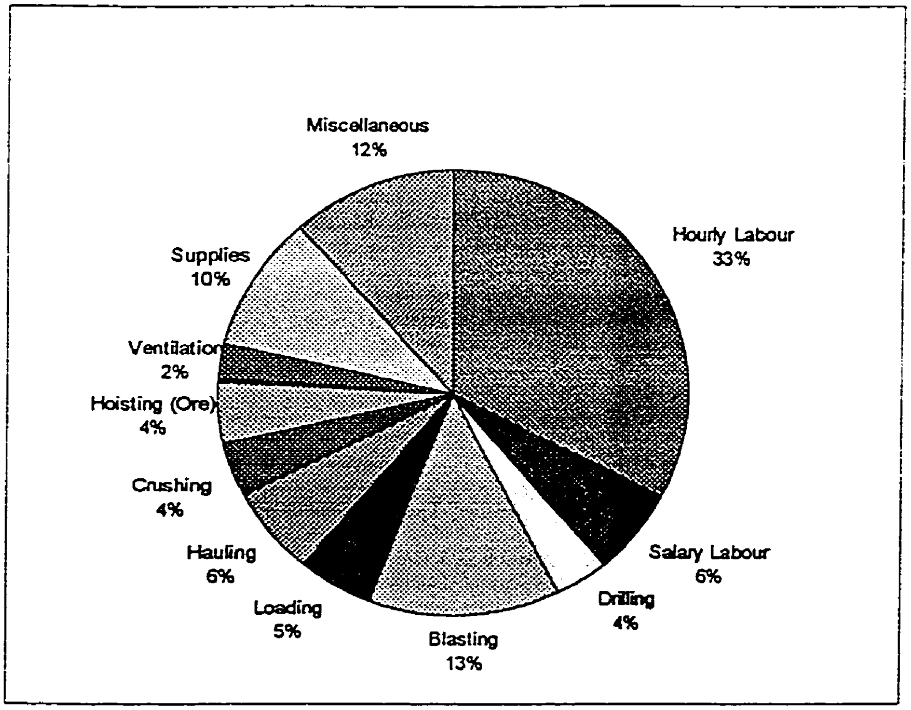

Figure 5.4.2.1. - Underground Operating Cost Break-down

5.5. Economic Comparison

Firstly, the open pit and underground cost flow profiles are determined according to equation 5.5.1. given below.

$$CF_i = OC_i(1 - t) + CE_i - (t \cdot DA_i) \qquad (5.5.1.)$$

where:	CFi	=	Cost Flow in year i
	OCi	=	Operating Cost in year i
	t	=	Tax rate
	CEi	=	Capital Expenditure in year i
	DAi	=	Depreciation Allowance in year i

Then, from the open pit cost flow profile, the present worth of future cost flows is determined as the reference point moves ahead one year at-at-time. Previous costs are considered sunk. In the case of the underground cost flow profile, the reference year is fixed at year 3, the start of underground production. However, to remain comparable to the remaining open pit mine life, the life of the underground operation is decreased accordingly. Preproduction capital costs are appreciated over the 2-year period.

A minimum underground life of 5 years is required to justify the comparison. This restriction, along with the underground development period of 2 years, limits the analysis to a period of 13 years.

An overall tax rate of 40 percent is applied in both cases. A 10-percent cost of capital is assumed.

Table 5.5.1. - Annual Open Pit Cost Flow Profile and Present Worth of Future Cost Flows

Open Pit																				
Year (US$ Million)	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Annual Operating Costs	5.42	5.42	5.42	5.42	5.42	6.58	6.58	6.58	6.58	6.58	7.85	7.85	7.85	7.85	7.85	7.85	7.85	7.85	7.85	7.85
Annual Capital Costs	1.36	0.00	0.00	2.34	0.56	4.13	0.00	0.00	1.32	0.03	2.43	0.56	0.56	1.62	0.31	0.93	2.03			
Depreciation	0.41	0.29	0.20	0.84	0.76	1.77	1.24	0.87	1.00	0.71	1.21	1.02	0.88	1.08	0.85	0.84	1.19	0.84	0.57	0.40
After Tax Cost Flow	4.45	3.14	3.17	5.25	3.51	7.37	3.45	3.60	4.86	3.69	6.65	4.87	4.92	5.90	4.68	5.31	6.26	4.38	4.48	4.55
Present Worth	38.46	37.86	38.51	39.18	37.85	38.12	34.56	34.57	34.43	33.00	32.62	29.22	27.28	25.08	21.69	19.18	15.79	11.11	7.84	4.14

Table 5.5.2. - Annual Underground Cost Flow Profile and Present Worth of Future Cost Flows

Underground																				
Year (US$ Million)	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Annual Operating Costs	0.00	0.00	5.15	5.15	5.15	5.15	5.15	5.15	5.15	5.15	5.15	5.15	5.15	5.15	5.15	5.15	5.15	5.15	5.15	5.15
Annual Capital Costs	5.48	13.03						1.06					1.96		0.42	0.80				
Depreciation	5.48	5.83	2.16	1.51	1.06	0.74	0.52	0.68	0.48	0.33	0.23	0.16	0.70	0.49	0.47	0.57	0.40	0.28	0.20	0.14
After Tax Cost Flow	3.29	10.70	2.23	2.48	2.67	2.79	2.88	3.88	2.90	2.96	3.00	3.02	4.77	2.89	3.32	3.66	2.93	2.98	3.01	3.03
Present Worth	-	-	38.59	38.05	37.45	36.80	36.10	35.14	34.18	33.26	31.58	30.42	29.15	27.77	26.28	24.09				

Table 5.5.1. gives the Open Pit cost flow profile as well as the present worth of future cost flows for each year of the 20-year planning period. Depreciation allowances are determined using the declining-balance method with an annual rate of 30 percent for all capital expenditures.

Table 5.5.2. gives the Underground cost flow profile and present worth of future cost flows for each year of the planning period. Development expenditures are expensed in the year in which they are incurred. Depreciation allowances for the equipment are determined using the declining-balance method with an annual rate of 30 percent.

Figure 5.5.1. shows the Open Pit and Underground Present Worths for each year.

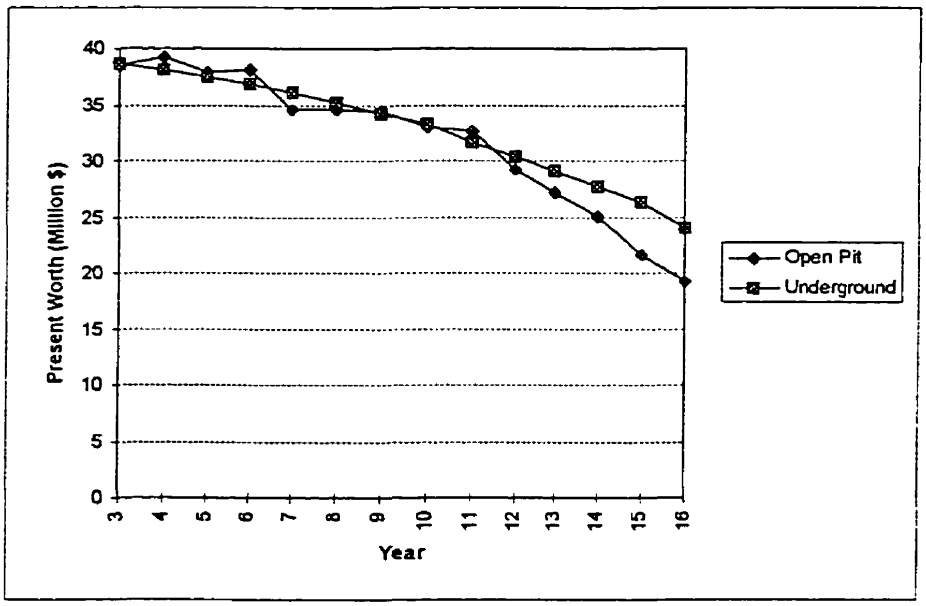

Figure 5.5.1. - Present Worth Analysis

Based on these results, the consultant reckons that the conversion from open pit to underground operations,should be made in year 6 because of the highest difference between Open Pit and Underground present worths (US$ 1.32 million), meaning that underground development should start at the beginning of year 4. However, before such a decision is recommended, the consultant decides to carry out a sensitivity analysis of the project.

5.6. Sensitivity Analysis

The sensitivity analysis will show the effect of changes in critical parameters on the difference between the present worth results of Open Pit and Underground mining options. Year 6 is the reference year of analysis. The parameters selected are:

- Long-term Open Pit operating costs;
- Underground operating costs;
- Discount rate;
- Tax rate.

Each one of the above parameters is changed one at a time within a range of -30/+30 percent. The results are shown in Figure 5.6.1. The results show that the operating cost for both alternatives is the most important parameter affecting the decision. The tax rate has almost no effect and it is not important if inexact value is used. Regarding the discount rate, it indicates at what value both alternatives are equally preferred.

Positive values mean that the open pit alternative is costlier than the underground alternative, since Figure 5.6.1. is based on the difference in present worth results. Hence, the underground option has a lower cost. Negative values indicate that the open pit alternative is preferred. As seen in Figure 5.6.1., the open pit alternative is more economic if the long-term open pit operating costs are at least 4 percent lower than estimated or if the underground operating costs are at least 6 percent higher.

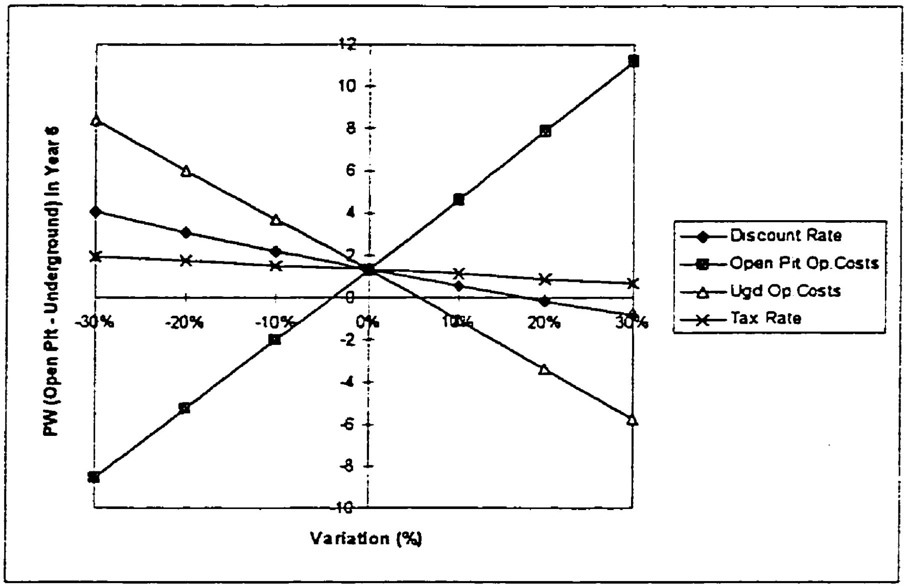

Figure 5.6.1. - Spider Diagram

5.7. Conclusion

Based on the base case and sensitivity analysis results, the consultant concludes that a conversion in year 6 is feasible. However, the consultant recommends to the management team that a detailed feasibility study be carried out before any final decision is taken.

CONCLUSION AND FUTURE PROSPECTS

6.1. Conclusion

The software developed for this thesis provides an easy-to-use tool to assist limestone mine operators in the analysis of underground conversion potential. If properly used, the program can be an essential tool for long-term planning, in addition to other planning programs already available. Its application enables a corporation to foresee years in advance the most favourable time for underground conversion, if any, and therefore, maximize the present worth of the operation. In this way, the corporation has enough time to take the necessary measures to investigate the issue further.

The break-even model also allows frequent analysis of different operating options in a short period of time. However, the software should not be used as the sole element on which to base the underground conversion decision. The results have a pre-feasibility study level accuracy and, needless to say, the reliability of the output is directly conditioned to the quality of the input data.

The current version of the software has some limitations. In terms of flexibility of data handling, there is scope for improvement. For instance, in case of any mistake during the data entry, the user must restart the program and enter all the data again. There is also a lack of flexibility for certain calculations, such as the tax system and depreciation method, in which no options are available.

The major shortcoming of the software, however, is related to the very nature of the cost functions used. O'Hara's equations were developed based mostly on metal mines and not specifically for limestone operations, which have particularities associated with the type of the deposit and the low unit value of the commodity. Therefore, the cost function estimates have a low degree of accuracy and should be used with this limitation in mind.

Despite its limitations and shortcomings, however, the software fully accomplishes the objectives of this thesis, providing an easy-to-use and reliable tool to assist the limestone mine operator in analysing underground conversion.

6.2. Future Prospects

The software can be improved by the consideration of the following aspects:

1. Update of O'Hara and Suboleski's cost estimation equations -- these regression-type equations were derived from actual mining costs collected during the '80s. Therefore, they do not account for technological advances in recent years. This work would involve a comprehensive worldwide mining cost survey of industrial mineral operations in order to obtain the required data. The updated equations could then be integrated into the software, improving its estimation accuracy.

2. Full cash-flow implementation -- to make the model suitable for metal mines, a revenue component must be factored in. In a metal mine, the revenue is a function of the grade of the ore and price of the product. The long-term plan would require annual grade estimates as well as price forecasts. Sensitivity analysis could be performed with respect to price forecasts as well.

3. Integration with a sequence optimization program -- one step further would be to integrate the software with a mine sequencing optimization program. This would allow the maximization of present worth of the whole project, combining the open pit and underground operations.

4. Implementation of inflation calculation -- to improve the accuracy of the results, the model has to take inflation into account. Therefore, the addition of an inflation adjustment function is highly recommended.

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```
/* Program to Compare Present Worths of Open Pit (OP) & Underground (UG)
Operations and Recommend the Best Economic Year to Switch from OP to UG (if any) */

# include <stdio.h>
# include <stdlib.h>
# include <conio.h>
# include <math.h>
# define LIFE 20
# define DELAY 150000

float otf, tax_rate, op_flow[LIFE], ug_flow[LIFE], dbd_rate, disc_rate;
int min_life;


int menu(void);
void delay(void);
void clear_kb(void);
void clrscr(void);
void op_user(float op_flow[]);
void ug_user(float ug_flow[]);
void op_model(float op_flow[]);
void ug_model(float ug_flow[]);
void calc(float op_flow[], float ug_flow[]);
float prescost(float x[], int y, int z, float k);

main()
{

        while (1)
        {
        /* Get user's selection and branch based on the input. */

                switch(menu())
                {
                        case 0:
                        {
                                clrscr();
                                puts ("** Cost-Flow Data Input **");
                                puts ("   ** General Data **");
                                puts (" ");
                                printf ("\n\nEnter Tax Rate (%):    ");
                                scanf ("%f",&tax_rate);
                                tax_rate=tax_rate/100;
                                otf=1-tax_rate;
                                printf ("\n\nEnter Declining Balance Depreciation Rate (%):  ");
                                scanf ("%f",&dbd_rate);
                                dbd_rate=dbd_rate/100;
```

```
        printf("\n\nEnter Discount Rate (%) = ");
        scanf("%f", &disc_rate);
        disc_rate=disc_rate/100;
        printf("\n\nEnter Minimum UGD Life (Years) = ");
        scanf("%d", &min_life);

        delay();
        break;
}

case 1:
{
        clrscr();
        op_user(op_flow);
        delay();
        break;
}

case 2:
{
        clrscr();
        ug_user(ug_flow);
        delay();
        break;
}

case 3:
{
        clrscr();
        op_model(op_flow);
        delay();
        break;
}

case 4:
{
        clrscr();
        ug_model(ug_flow);
        delay();
        break;
}

case 5:
{
        clrscr();
        calc(op_flow, ug_flow);
        delay();
        break;
}

case 6:  /* Exit program. */
{
        puts("\n Exiting program now...");
        delay();
```

```
                                exit(0);
                        }

                        default:
                        {
                                clrscr();
                                puts(" Invalid choice, try again.");
                                delay();
                        }
                }
        }
}

int menu(void)
/* Displays a menu and inputs user"s selection. */
{

        int reply;

        clrscr();
        puts("\n\tEnter 0 for GENERAL DATA ENTRY");
        puts("\n\tEnter 1 for OPEN PIT DATA ENTRY - USER'S ESTIMATION.");
        puts("\n\tEnter 2 for UNDERGROUND DATA ENTRY - USER'S ESTIMATION.");
        puts("\n\tEnter 3 for OPEN PIT DATA ENTRY - MODEL ESTIMATION.");
        puts("\n\tEnter 4 for UNDERGROUND DATA ENTRY - MODEL ESTIMATION.");
        puts("\n\tEnter 5 for COST-FLOW & BREAK-EVEN CALCULATIONS.");
        puts("\n\tEnter 6 to EXIT");

        scanf("%d", &reply);

        return reply;

}

/* Function to Delay Screen Display */

void delay(void)
{
        long x;
        for (x=0; x<DELAY; x++)
                ;
}

/* ==================================================================*/
/* Open Pit Data Entry Function - User's Estimation */
/* ==================================================================*/

void op_user(float op_flow[])
{
        float ore_prod [LIFE], strip_ratio[LIFE];
        float oper_cost[LIFE], cap_ex[LIFE], cap_life[LIFE], env_cost;
        float tax_cred[LIFE], oper_tot[LIFE];
        int time, temp;
```

```
char filename[20];
int reply;

FILE *fp;

puts ("Read a Data File? Yes=2  No=1");
scanf("%d",&reply);

switch (reply)

{
        case 1:

        {
                /* Save Data in an ASCII file*/
                clear_kb();
                puts ("Save the Data in an ASCII file.");
                puts ("Enter file name with extension.");
                gets (filename);

                if ((fp=fopen(filename,"w")) == NULL)

                {
                        fprintf(stderr, "Error opening file %s.", filename);
                        exit(1);
                }

                clrscr();

                puts ("** Cost-Flow Data Input **");
                puts ("   ** Open Pit Data **");
                puts (" ");


                /* Data Input */

                printf ("Enter Environmental Cost per Ton: ");
                scanf ("%f",&env_cost);

                for (time=1; time<LIFE; time++)
                {
                        gotoxy (6,4);
                        printf ("Data Entry for Year %d:",time);
                        puts (" ");
                        printf ("Enter ROM (short tons) for year %d:  ",time);
                        scanf ("%f",&ore_prod[time]);
                        fprintf(fp,"\n%.2f",ore_prod[time]);
                        printf ("Enter Stripping Ratio (Waste/Ore) for year %d:  ",time);
                        scanf ("%f",&strip_ratio[time]);
                        fprintf(fp,"\t%.2f",strip_ratio[time]);
                        printf ("Enter Operating Mining Cost (US$/ton) for year %d:
",time);

                        scanf ("%f",&oper_cost[time]);
                        fprintf(fp,"\t%.2f",oper_cost[time]);
```

```
                        printf ("Enter Capital Expenditures (US$ million) for year %d:
",time);
                        scanf ("%f",&cap_ex[time]);
                        fprintf(fp,"\t%.2f",cap_ex[time]);
                        printf ("Enter Expected Life (years) for above Capex:  ");
                        scanf ("%f",&cap_life[time]);
                        fprintf(fp,"\t%.2f",cap_life[time]);

                        clrscr();

                }

                fclose(fp);

                break;

        }

        case 2:

        {
                clear_kb();
                puts ("Read Data from an ASCII file.");
                puts ("Enter file name with extension.");
                gets (filename);

                if ((fp = fopen(filename,"r")) == NULL)
                {
                        fprintf(stderr, "Error opening file.");
                        exit(1);
                }

                for (time=1; time<LIFE; time++)
                {
                        fscanf(fp,"%f",&ore_prod[time]);

                        fscanf(fp,"%f",&strip_ratio[time]);

                        fscanf(fp,"%f",&oper_cost[time]);

                        fscanf(fp,"%f",&cap_ex[time]);

                        fscanf(fp,"%f",&cap_life[time]);

                }

                fclose(fp);

        }

}

/* Tax Credit Calculation -  Declining Balance Depreciation @ rate */
```

```
	for (time=1; time<LIFE; time++)

	{
		for (temp=time+1; temp<=(cap_life[time]+time); temp++)
		{
			tax_cred[temp]=cap_ex[time]*pow((1-dbd_rate),(temp-time-1))*dbd_rate+
tax_cred[temp];
		}

		tax_cred[temp]=tax_cred[temp]*tax_rate;

	}

	/* Cost Flow Calculation */

	for (time=1; time<LIFE; time++)

	{
		oper_cost[time]=oper_cost[time]+env_cost;
		oper_tot[time]=ore_prod[time]*(1+strip_ratio[time])*oper_cost[time];
		op_flow[time]=oper_tot[time]*otf+cap_ex[time]-tax_cred[time];
		op_flow[time]=op_flow[time]/1000000;

	}

	/* Save Results in ASCII file*/
	clrscr();
	clear_kb();
	puts ("Save Results in ASCII file.");
	puts ("Enter file name with extension.");
	gets (filename);

	if ((fp=fopen(filename,"w")) == NULL)

	{
		fprintf(stderr, "Error opening file %s.", filename);
		exit(1);
	}

	for (time=1; time<LIFE; time++)
	{
		fprintf (fp,"\n%d",time);
		fprintf (fp,"\t%.2f",op_flow[time]);
	}

	fclose(fp);
}


void clear_kb(void)
/* Clears stdin of any waiting characters.*/

{
	char junk[80];
```

```
        gets(junk);
}

/* ==============================================================*/
/* Underground Data Entry Function - User's Estimation */
/* ==============================================================*/

void ug_user(float ug_flow[])

{
        float ore_prod , capex_perc[2];
        float oper_cost;
        float tax_cred[LIFE], ugd_dev, ugd_equip, ugd_capex;
        float ugd_opcost;
        int time, temp, power;

        char filename[20];
        int reply;

        FILE *fp;

        puts ("Read a Data File? Yes=2  No=1");
        scanf("%d",&reply);

        switch (reply)
        {
                case 1:

                {
                        /* Save Data in an ASCII file*/
                        clear_kb();
                        puts ("Save the Data in an ASCII file.");
                        puts ("Enter file name with extension.");
                        gets (filename);

                        if ((fp=fopen(filename,"w")) == NULL)

                        {
                                fprintf(stderr, "Error opening file %s.", filename);
                                exit(1);
                        }

                        puts ("** Cost-Flow Data Input **");
                        puts ("   ** Underground ** ");
                        puts (" ");

                        /* Data Input */

                        gotoxy (6,4);
                        printf ("Pre-Production Data Entry");
                        puts (" ");

                        printf ("Enter Pre-Production Development Cost - US$ Millions");
                        scanf ("%f",&ugd_dev);
```

```
                fprintf(fp,"\n%.2f",ugd_dev);

                printf ("Enter Pre-Production Equipment Cost - US$ Millions");
                scanf ("%f",&ugd_equip);
                fprintf(fp,"\n%.2f",ugd_equip);

                printf ("Enter Annual Production (short tons)");
                scanf ("%f",&ore_prod);
                fprintf(fp,"\n%.2f",ore_prod);

                printf ("Enter Operating Mining Cost (US$/ton)");
                scanf ("%f",&oper_cost);
                fprintf(fp,"\n%.2f",oper_cost);

                clrscr();

                fclose(fp);

                break;

        }

        case 2 :

        {
                clear_kb();
                puts ("Read Data from an ASCII file.");
                puts ("Enter file name with extension.");
                gets (filename);

                if ((fp = fopen(filename,"r")) == NULL)
                {
                        fprintf(stderr, "Error opening file.");
                        exit(1);
                }

                fscanf(fp,"%f",&ugd_dev);

                fscanf(fp,"%f",&ugd_equip);

                fscanf(fp,"%f",&ore_prod);

                fscanf(fp,"%f",&oper_cost);

                fclose(fp);

        }

}

        ugd_capex = ugd_dev + ugd_equip;

/* Tax Credit Calculation */
/* Depreciation on Development @ 100% & Equipment DBD @ rate */
```

```
/* Equipment Depreciation starts only after pre-production */

for (temp=3; temp<=LIFE; temp++)
{
        power=temp-3;
        tax_cred[temp]=ugd_equip * pow((1-dbd_rate),power) * dbd_rate;
        tax_cred[temp]= tax_cred[temp]*tax_rate;
}

/* Development Depreciation splitted over 2 years */

tax_cred[1]= 0.5*ugd_dev*tax_rate;
tax_cred[2]= 0.5*ugd_dev*tax_rate;


/* Cost Flow Calculation */


/* Splits Capex over 2-year pre-production */

printf("\nEnter Percentage of Capex in Year 1 = ");
scanf ("%f",&capex_perc[1]);

printf("\nEnter Percentage of Capex in Year 2 = ");
scanf ("%f",&capex_perc[2]);

ug_flow[0]=0;

ug_flow[1]=capex_perc[1]*ugd_capex/100;

ug_flow[1]=(ug_flow[1]-tax_cred[1])/1000000;

ug_flow[2]=capex_perc[2]*ugd_capex/100;

ug_flow[2]=(ug_flow[2]-tax_cred[2])/1000000;


/* Annual Operating Costs */

ugd_opcost = ore_prod * oper_cost;

for (time=3; time<=LIFE; time++)
{
        ug_flow[time]=ugd_opcost*otf - tax_cred[time];
        ug_flow[time]=ug_flow[time]/1000000;
}

/* Save Results in ASCII file*/
clrscr();
clear_kb();
puts ("Save Results in ASCII file.");
puts ("Enter file name with extension.");
gets (filename);
```

```
	if ((fp=fopen(filename,"w")) == NULL)

	{
		fprintf(stderr, "Error opening file %s.", filename);
		exit(1);
	}

	for (time=1; time<LIFE; time++)
	{
		fprintf (fp,"\n%d",time);
		fprintf (fp,"\t%.2f",ug_flow[time]);
	}

	fclose(fp);

}

/* ================================================================*/
/* Open Pit Data Entry Function - Model Estimation */
/* Open Pit Cost Estimation based on O'Hara & Suboleski*/
/* ================================================================*/

void op_model(float op_flow[])
{
	float prod_rate, index, factor, strip_ratio[LIFE], daily[LIFE],capex[LIFE];
	float size, prod, drill_cost[LIFE], blast_cost[LIFE], load_cost[LIFE];
	float haul_cost[LIFE], serv_cost[LIFE], crush_cost[LIFE], oper_cost[LIFE];
	float op_factor, capex_load[LIFE], env_ton, env_cost[LIFE], power, tax_cred[LIFE];
	float capex_truck[LIFE];

	int days_year, time, temp, number,  time_left;
	int add_drill, equip_life, year, equip[LIFE];
	int equip_req[LIFE], replace, old_equip;
	int equip_add[LIFE], new_repl[LIFE];

	char filename[20];
	int reply;

	FILE *fp;

	puts ("Read a Data File? Yes=2  No=1");
	scanf("%d",&reply);

	switch (reply)

	{
		case 1:

		{
			/* Save Data in an ASCII file*/
			clear_kb();
			puts ("Save the Data in an ASCII file.");
			puts ("Enter file name with extension.");
			gets (filename);
```

```
if ((fp=fopen(filename,"w")) == NULL)

{
        fprintf(stderr, "Error opening file %s.", filename);
        exit(1);
}

clrscr();

puts ("** Cost-Flow Data Input **");
puts ("   ** Open Pit Data **");
puts (" ");


printf("\nEnter Annual Production Rate (short tons) = ");
scanf ("%f",&prod_rate);
fprintf(fp,"\n%.2f",prod_rate);

printf("\nEnter Days of Operation per Year = ");
scanf ("%d",&days_year);
fprintf(fp,"\t%d",days_year);

printf("\nEnter Marshall&Swift Index for Base Year  = ");
scanf ("%f",&index);
fprintf(fp,"\t%.2f",index);

printf("\nEnter Operating Index for Base Year  = ");
scanf ("%f",&op_factor);
fprintf(fp,"\t%.2f",op_factor);

op_factor=op_factor/873.9;

factor=index/873.9;

printf("\nEnter Environmental Cost per ton = ");
scanf ("%f",&env_ton);
fprintf(fp,"\t%.2f",env_ton);

/* SR Entry and Daily Tonnage Calculation */

for (time=1; time<=LIFE; time++)
{
        printf("\nEnter Stripping Ratio (W/O) for Year %d = ",time);
        scanf ("%f",&strip_ratio[time]);
        fprintf(fp,"\n%.2f",strip_ratio[time]);

        daily[time]=prod_rate*(1+strip_ratio[time])/days_year;
}

/* Environmental Costs per Year */

for (time=1; time<=LIFE; time++)
{
```

```
env_cost[time]=prod_rate * (1+strip_ratio[time])* env_ton;

}

prod=prod_rate/days_year;

clrscr();

/* Drill Data Entry */

printf("\nEnter Number of Drills = ");
scanf ("%d",&number);
fprintf(fp,"\n%d",number);

printf("\nEnter Diameter of Drillholes (inches) = ");
scanf ("%f",&size);
fprintf(fp,"\t%.2f",size);

printf("\nEnter Remaining Economic Life = ");
scanf ("%d",&time_left);
fprintf(fp,"\t%d",time_left);

printf("\nEnter Year of Additional Drill Acquisition = ");
scanf ("%d",&add_drill);
fprintf(fp,"\t%d",&add_drill);

printf("\nEnter Drills Economic Life = ");
scanf ("%d",&equip_life);
fprintf(fp,"\t%d",equip_life);

/* Replacement of Existing Drills */

replace=floor((LIFE-time_left)/equip_life);


for (time=0; time<=LIFE; time++)
{
        equip[time]=0;
        capex[time]=0;
}

for (time=0; time<=replace; time++)
{
        year=time_left+time*equip_life;

        equip[year]=number;
}

/* Depreciation on Existing Drills */
/* Declining Balance Method @ rate */

old_equip=number*20000*pow(size,1.8)*factor;

for (temp=1; temp<=time_left; temp++)
```

```
			{
				power=equip_life-time_left+temp-1;

				tax_cred[temp]=old_equip*pow((1-dbd_rate),power)*dbd_rate+
tax_cred[temp];
			}

			/* Replacement of Additional Drill */

			replace=floor((LIFE-add_drill)/equip_life);


			for (time=0; time<=replace; time++)
			{
				year=add_drill+time*equip_life;

				equip[year]=equip[year]+1;
			}

			for (time=1; time<=LIFE; time++)
			{
				capex[time]=equip[time]*20000*pow(size,1.8)*factor;

				/* Depreciation on New Drills */

				if (equip[time]>0)
				{
					for (temp=time+1; temp<=(equip_life+time); temp++)

					tax_cred[temp]=capex[time]*pow((1-dbd_rate),(temp-time-
1))*dbd_rate+ tax_cred[temp];
				}
			}

			clrscr();

	/* Loader Data Entry */

	printf("\nEnter Number of Loaders = ");
	scanf ("%d",&number);
	fprintf(fp,"\n%d",number);

	printf("\nEnter Loaders Capacity (yd3) = ");
	scanf ("%f",&size);
	fprintf(fp,"\t%.2f",size);

	printf("\nEnter Remaining Economic Life = ");
	scanf ("%d",&time_left);
	fprintf(fp,"\t%d",time_left);

	printf("\nEnter Loader Economic Life = ");
	scanf ("%d",&equip_life);
	fprintf(fp,"\t%d",equip_life);
```

```
/* Replacement of Existing Loaders */

replace=floor((LIFE-time_left)/equip_life);


for (time=0; time<=LIFE; time++)
{
        equip[time]=0;
}

for (time=0; time<=replace; time++)
{
        year=time_left+time*equip_life;

        equip[year]=number;
}

/* Depreciation on Existing Loaders */

old_equip=number*510000*pow(size,0.8)*factor;

for (temp=1; temp<=time_left; temp++)
        {
                power=equip_life-time_left+temp-1;

                tax_cred[temp]=old_equip*pow((1-dbd_rate),power)*dbd_rate+
tax_cred[temp];
        }

/* Number of Loaders Required & Additional*/

equip_req[0]=number;

for (time=1; time<=LIFE; time++)
{
        equip_req[time]=floor(0.011*pow(daily[time],0.8)/size);

        equip_add[time]=equip_req[time]-equip_req[time-1];

        if (equip_add[time]>0)
        {
                replace=floor((LIFE-time)/equip_life);

                for (temp=1; temp<=replace; temp++)
                {
                        year=time+temp*equip_life;

                        new_repl[year]=new_repl[year]+equip_add[time];
                }
        }


        equip[time]=equip[time]+equip_add[time]+new_repl[time];
}
```

```
equip[LIFE]=0;

/* Loaders Capital Cost */

for (time=1; time<=LIFE; time++)
{
        capex_load[time]=equip[time]*510000*pow(size,0.8)*factor;
        capex[time]=capex[time]+ capex_load[time];

        /* Depreciation on New Loaders */

        if (equip[time]>0)
        {
                for (temp=time+1; temp<=(equip_life+time); temp++)

                tax_cred[temp]=capex_load[time]*pow((1-dbd_rate),(temp-time-
1))*dbd_rate+ tax_cred[temp];
        }
}
clrscr();

/* Truck Data Entry */

printf("\nEnter Number of Trucks = ");
scanf ("%d",&number);
fprintf(fp,"\n%d",number);

printf("\nEnter Truck Capacity (st) = ");
scanf ("%f",&size);
fprintf(fp,"\t%.2f",size);

printf("\nEnter Remaining Economic Life = ");
scanf ("%d",&time_left);
fprintf(fp,"\t%d",time_left);

printf("\nEnter Truck Economic Life = ");
scanf ("%d",&equip_life);
fprintf(fp,"\t%d",equip_life);


/* Replacement of Existing Trucks */

replace=floor((LIFE-time_left)/equip_life);


for (time=0; time<=LIFE; time++)
{
        equip[time]=0;
        new_repl[time]=0;
}

for (time=0; time<=replace; time++)
{
```

```
            year=time_left+time*equip_life;

            equip[year]=number;
    }

    /* Depreciation on Existing Trucks */

    old_equip=number*20400*pow(size,0.9)*factor;

    for (temp=1; temp<=time_left; temp++)
            {
                    power=equip_life-time_left+temp-1;

                    tax_cred[temp]=old_equip*pow((1-dbd_rate),power)*dbd_rate+
tax_cred[temp];
            }

    /* Number of Trucks Required & Additional*/

    equip_req[0]=number;

    for (time=1; time<=LIFE; time++)
    {
            equip_req[time]=floor(0.25*pow(daily[time],0.8)/size);

            equip_add[time]=equip_req[time]-equip_req[time-1];

            if (equip_add[time]>0)
            {
                    replace=floor((LIFE-time)/equip_life);

                    for (temp=1; temp<=replace; temp++)
                    {
                            year=time+temp*equip_life;

                            new_repl[year]=new_repl[year]+equip_add[time];

                    }
            }

            equip[time]=equip[time]+new_repl[time]+equip_add[time];
    }

    equip[LIFE]=0;

    /* Trucks Capital Cost */

    for (time=1; time<=LIFE; time++)
    {
            capex_truck[time] = equip[time]*20400* pow(size,0.9)*factor;
            capex[time]=capex[time] + capex_truck[time];

            /* Depreciation on New Trucks */
```

```
		if (equip[time]>0)
		{
			for (temp=time+1; temp<=(equip_life+time); temp++)

			tax_cred[temp]=capex_truck[time]*pow((1-dbd_rate),(temp-time-
1))*dbd_rate+ tax_cred[temp];
		}
	}


	capex[LIFE]=0;

		fclose(fp);

			break;

	}

	case 2 :

	{
			clear_kb();
			puts ("Read Data from an ASCII file.");
			puts ("Enter file name with extension.");
			gets (filename);

			if ((fp = fopen(filename,"r")) == NULL)
			{
				fprintf(stderr, "Error opening file.");
				exit(1);
			}

			fscanf (fp,"%f",&prod_rate);

			fscanf (fp,"%d",&days_year);

			fscanf (fp,"%f",&index);

			fscanf (fp,"%f",&op_factor);


			op_factor=op_factor/873.9;

			factor=index/873.9;


			fscanf (fp,"%f",&env_ton);


			/* SR Entry and Daily Tonnage Calculation */

			for (time=1; time<=LIFE; time++)
			{
				fscanf (fp,"%f",&strip_ratio[time]);
```

```
        daily[time]=prod_rate*(1+strip_ratio[time])/days_year;
}

/* Environmental Costs per Year */

for (time=1; time<=LIFE; time++)
{
env_cost[time]=prod_rate * (1+strip_ratio[time])* env_ton;

}

prod=prod_rate/days_year;

clrscr();

/* Drill Data Entry */

fscanf (fp,"%d",&number);

fscanf (fp,"%f",&size);

fscanf (fp,"%d",&time_left);

fscanf (fp,"%d",&add_drill);

fscanf (fp,"%d",&equip_life);


/* Replacement of Existing Drills */

replace=floor((LIFE-time_left)/equip_life);


for (time=0; time<=LIFE; time++)
{
        equip[time]=0;
        capex[time]=0;
}

for (time=0; time<=replace; time++)
{
        year=time_left+time*equip_life;

        equip[year]=number;
}

/* Depreciation on Existing Drills */
/* Declining Balance Method @ rate */

old_equip=number*20000*pow(size,1.8)*factor;

for (temp=1; temp<=time_left; temp++)
{
```

```
                    power=equip_life-time_left+temp-1;

                    tax_cred[temp]=old_equip*pow((1-dbd_rate),power)*dbd_rate+
tax_cred[temp];
            }

            /* Replacement of Additional Drill */

            replace=floor((LIFE-add_drill)/equip_life);


            for (time=0; time<=replace; time++)
            {
                    year=add_drill+time*equip_life;

                    equip[year]=equip[year]+1;
            }

            for (time=1; time<=LIFE; time++)
            {
                    capex[time]=equip[time]*20000*pow(size,1.8)*factor;

                    /* Depreciation on New Drills */

                    if (equip[time]>0)
                    {
                            for (temp=time+1; temp<=(equip_life+time); temp++)

                            tax_cred[temp]=capex[time]*pow((1-dbd_rate),(temp-time-
1))*dbd_rate+ tax_cred[temp];
                    }
            }

            clrscr();

    /* Loader Data Entry */

    fscanf (fp,"%d",&number);

    fscanf (fp,"%f",&size);

    fscanf (fp,"%d",&time_left);

    fscanf (fp,"%d",&equip_life);


    /* Replacement of Existing Loaders */

    replace=floor((LIFE-time_left)/equip_life);


    for (time=0; time<=LIFE; time++)
    {
            equip[time]=0;
```

```
	}

	for (time=0; time<=replace; time++)
	{
		year=time_left+time*equip_life;

		equip[year]=number;
	}

	/* Depreciation on Existing Loaders */

	old_equip=number*510000*pow(size,0.8)*factor;

	for (temp=1; temp<=time_left; temp++)
		{
			power=equip_life-time_left+temp-1;

			tax_cred[temp]=old_equip*pow((1-dbd_rate),power)*dbd_rate+
tax_cred[temp];
		}

 /* Number of Loaders Required & Additional*/

	equip_req[0]=number;

	for (time=1; time<=LIFE; time++)
	{
		equip_req[time]=floor(0.011*pow(daily[time],0.8)/size);

		equip_add[time]=equip_req[time]-equip_req[time-1];

		if (equip_add[time]>0)
		{
			replace=floor((LIFE-time)/equip_life);

			for (temp=1; temp<=replace; temp++)
			{
				year=time+temp*equip_life;

				new_repl[year]=new_repl[year]+equip_add[time];
			}
		}


		equip[time]=equip[time]+equip_add[time]+new_repl[time];
	}

	equip[LIFE]=0;

	/* Loaders Capital Cost */

	for (time=1; time<=LIFE; time++)
	{
		capex_load[time]=equip[time]*510000*pow(size,0.8)*factor;
```

```
                capex[time]=capex[time]+ capex_load[time];

                /* Depreciation on New Loaders */

                if (equip[time]>0)
                {
                        for (temp=time+1; temp<=(equip_life+time); temp++)

                        tax_cred[temp]=capex_load[time]*pow((1-dbd_rate),(temp-time-
1))*dbd_rate+ tax_cred[temp];
                }
        }
        clrscr();

        /* Truck Data Entry */

        fscanf (fp,"%d",&number);

        fscanf (fp,"%f",&size);

        fscanf (fp,"%d",&time_left);

        fscanf (fp,"%d",&equip_life);


        /* Replacement of Existing Trucks */

        replace=floor((LIFE-time_left)/equip_life);


        for (time=0; time<=LIFE; time++)
        {
                equip[time]=0;
                new_repl[time]=0;
        }

        for (time=0; time<=replace; time++)
        {
                year=time_left+time*equip_life;

                equip[year]=number;
        }

        /* Depreciation on Existing Trucks */

        old_equip=number*20400*pow(size,0.9)*factor;

        for (temp=1; temp<=time_left; temp++)
                {
                        power=equip_life-time_left+temp-1;

                        tax_cred[temp]=old_equip*pow((1-dbd_rate),power)*dbd_rate+
tax_cred[temp];
                }
```

```
/* Number of Trucks Required & Additional*/

equip_req[0]=number;

for (time=1; time<=LIFE; time++)
{
        equip_req[time]=floor(0.25*pow(daily[time],0.8)/size);

        equip_add[time]=equip_req[time]-equip_req[time-1];

        if (equip_add[time]>0)
        {
                replace=floor((LIFE-time)/equip_life);

                for (temp=1; temp<=replace; temp++)
                {
                        year=time+temp*equip_life;

                        new_repl[year]=new_repl[year]+equip_add[time];

                }
        }

        equip[time]=equip[time]+new_repl[time]+equip_add[time];
}

equip[LIFE]=0;

/* Trucks Capital Cost */

for (time=1; time<=LIFE; time++)
{
        capex_truck[time] = equip[time]*20400* pow(size,0.9)*factor;
        capex[time]=capex[time] + capex_truck[time];

        /* Depreciation on New Trucks */

        if (equip[time]>0)
        {
                for (temp=time+1; temp<=(equip_life+time); temp++)

                tax_cred[temp]=capex_truck[time]*pow((1-dbd_rate),(temp-time-
1))*dbd_rate+ tax_cred[temp];
        }
}


capex[LIFE]=0;

fclose(fp);


}
```

```
}
        /* Operating Costs*/

        for (time=1; time<=LIFE; time++)
        {
                drill_cost[time]=1.90*pow(daily[time],0.7)*op_factor;

                blast_cost[time]=3.17*pow(daily[time],0.7)*op_factor;

                load_cost[time]=2.67*pow(daily[time],0.7)*op_factor;

                haul_cost[time]=18.07*pow(daily[time],0.6)*op_factor;

                serv_cost[time]=6.65*pow(daily[time],0.7)*op_factor;

                crush_cost[time]=7.90*pow(prod,0.6)*op_factor;

                oper_cost[time]=drill_cost[time]+blast_cost[time]+load_cost[time];

                oper_cost[time]=oper_cost[time]+haul_cost[time]+serv_cost[time]+crush_cost[time];

                oper_cost[time]=days_year*oper_cost[time];

}

        /* Tax Credit Calculation */

        for (time=1; time<=LIFE; time++)
        {
                tax_cred[time]=tax_cred[time]*tax_rate;
        }

        /* Cost Flow Calculation */

        for (time=1; time<=LIFE; time++)
        {
         op_flow[time]= oper_cost[time]*otf+ capex[time]- tax_cred[time];
         op_flow[time]= (env_cost[time] + op_flow[time])/1000000;
        }

        /* Save Results in ASCII file*/
        clrscr();
        clear_kb();
        puts ("Save Results in ASCII file.");
        puts ("Enter file name with extension.");
        gets (filename);

        if ((fp=fopen(filename,"w")) == NULL)

        {
                fprintf(stderr, "Error opening file %s.", filename);
                exit(1);
        }
```

```
        for (time=1; time<LIFE; time++)
        {
                fprintf (fp,"\n%d",time);
                fprintf (fp,"\t%.2f",op_flow[time]);
        }

        fclose(fp);

}

/* ==========================================================*/
/* Underground Data Entry Function - Model Estimation */
/* Estimation based on O'Hara & Suboleski*/
/* ==========================================================*/

void ug_model(float ug_flow[])
{
        float prod_rate, daily, op_factor, capex_factor, shaft_diam, shaft_depth;
        float shaft_costfix, shaft_costdep, rope_speed, hoist_drum, horse_power;
        float hoist_area, frame_height, frame_weight, hoist_cost, hoist_install;
        float hoist_room, frame_cost, bin_cost, drift_leng, drift_area, ramp_leng;
        float ramp_area, serv_leng, serv_area, drift_8x8, ramp_8x8, serv_8x8;
        float develop_cost, stope_height, equip_cost, fan_hp, vent_cost, pump_hp;
        float pump_cost, water_cost, jaw_cost, jaw_inst, shop_cost, air_need;
        float air_cost, lat_ext, pipe_cost, elect_load, elect_cost, ugd_capex;
        float capex_perc[LIFE], correc_fac, rp_opcost, crush_opcost;
        float hoist_opcost, ugd_opcost, ugd_dev, ugd_equip, tax_cred[LIFE] ;

        int days_year, time, temp;

        char filename[20];
        int reply;

        FILE *fp;

        puts ("Read a Data File? Yes=2  No=1");
        scanf("%d",&reply);

        switch (reply)

        {
                case 1:

                {
                        /* Save Data in an ASCII file*/
                        clear_kb();
                        puts ("Save the Data in an ASCII file.");
                        puts ("Enter file name with extension.");
                        gets (filename);

                        if ((fp=fopen(filename,"w")) == NULL)

                        {
                                fprintf(stderr, "Error opening file %s.", filename);
```

```
                        exit(1);
                }

                clrscr();

                puts ("** Cost-Flow Data Input **");
                puts ("   ** Underground Data **");
                puts (" ");

printf("\nEnter Annual Production Rate (short tons) = ");
scanf ("%f",&prod_rate);
fprintf(fp,"\n%.2f",prod_rate);

printf("\nEnter Working Days per Year (days) = ");
scanf ("%d",&days_year);
fprintf(fp,"\t%d",days_year);

printf("\nEnter Capex Adjustment Factor = ");
scanf ("%f",&capex_factor);
fprintf(fp,"\t%.2f",capex_factor);

printf("\nEnter Operating Cost Adjustment Factor = ");
scanf ("%f",&op_factor);
fprintf(fp,"\t%.2f",op_factor);

/* Capital Cost of Shaft Sinking */

printf("\nEnter Circular Shaft Diameter (ft)  = ");
scanf ("%f",&shaft_diam);
fprintf(fp,"\n%.2f",shaft_diam);

printf("\nEnter Shaft Depth (ft) = ");
scanf ("%f",&shaft_depth);
fprintf(fp,"\t%.2f",shaft_depth);

/* Mine Development Costs - Stope and Pillar */

clrscr();

printf("\nEnter Total Drifts Length (ft)  = ");
scanf ("%f",&drift_leng);
fprintf(fp,"\n%.2f",drift_leng);

printf("\nEnter Drifts Cross-Sectional Area (ft2)  = ");
scanf ("%f",&drift_area);
fprintf(fp,"\t%.2f",drift_area);

printf("\nEnter Total Inclined Ramps Length (ft)  = ");
scanf ("%f",&ramp_leng);
fprintf(fp,"\t%.2f",ramp_leng);

printf("\nEnter Inclined Ramps Cross-Sectional Area (ft2)  = ");
scanf ("%f",&ramp_area);
fprintf(fp,"\t%.2f",ramp_area);
```

```
printf("\nEnter Total Service Excavations Length (ft)  = ");
scanf ("%f",&serv_leng);
fprintf(fp,"\t%.2f",serv_leng);

printf("\nEnter Service Excavations Cross-Sectional Area (ft2)  = ");
scanf ("%f",&serv_area);
fprintf(fp,"\t%.2f",serv_area);

printf("\nEnter Stope Width (ft) = ");
scanf ("%f",&stope_height);
fprintf(fp,"\n%.2f",stope_height);

clrscr();

printf("\nEnter Percentage of Capex in Year 1 = ");
scanf ("%f",&capex_perc[1]);
fprintf(fp,"\n%.2f",capex_perc[1]);

printf("\nEnter Percentage of Capex in Year 2 = ");
scanf ("%f",&capex_perc[2]);
fprintf(fp,"\t%.2f",capex_perc[2]);


fclose(fp);

break;

        }

case 2 :

        {
                clear_kb();
                puts ("Read Data from an ASCII file.");
                puts ("Enter file name with extension.");
                gets (filename);

                if ((fp = fopen(filename,"r")) == NULL)
                {
                        fprintf(stderr, "Error opening file.");
                        exit(1);
                }

                fscanf (fp,"%f",&prod_rate);

                fscanf (fp,"%d",&days_year);

                fscanf (fp,"%f",&capex_factor);

                fscanf (fp,"%f",&op_factor);

                fscanf (fp,"%f",&shaft_diam);
```

```
                    fscanf (fp,"%f",&shaft_depth);

                    fscanf (fp,"%f",&drift_leng);

                    fscanf (fp,"%f",&drift_area);

                    fscanf (fp,"%f",&ramp_leng);

                    fscanf (fp,"%f",&ramp_area);

                    fscanf (fp,"%f",&serv_leng);

                    fscanf (fp,"%f",&serv_area);

                    fscanf (fp,"%f",&capex_perc[1]);

                    fscanf (fp,"%f",&capex_perc[2]);

                    fscanf (fp,"%f",&stope_height);

                    fclose(fp);

          }
}

daily=prod_rate/days_year;

/* Shaft Sinking - Circular */

shaft_costfix=135000*pow(shaft_diam,0.5);

shaft_costdep=307*pow(shaft_diam,0.7)*pow(shaft_depth,1.05);


/* Capital Costs of Hoisting Plant */
/* Double Drum Hoist < 1500 tpd* /

/* Rope Speed in fpm */

rope_speed=1.6*pow(shaft_depth,0.5)*pow(daily,0.4);

/* Hoist Drum Diameter in inches */

hoist_drum= 4.13*pow(daily,0.3)*pow(shaft_depth,0.14);

/* Horsepower in Hp */

horse_power=0.5*pow((hoist_drum/100),2.4)*rope_speed;

/* Hoistroom Area in ft */

hoist_area=0.10*pow(hoist_drum,2.2);

/* Headframe Size (ft) & Weight (lbs)*/
```

```
frame_height=8.0*pow(daily,0.3)+1.2*pow(rope_speed,0.5);

frame_weight=0.12*pow(frame_height,3)*pow((hoist_drum/100),2);

/* Cost of Hoist */

hoist_cost=700*pow((0.9*hoist_drum),1.4)*pow(horse_power,0.2);

hoist_install=64*pow(hoist_drum,1.8);

hoist_room=4.90*pow(hoist_area,1.4);

/* Capital Cost of Single-hoist Headframe */

frame_cost=19*pow(frame_weight,0.9);

/* Cost of Ore bins & skips etc. */

bin_cost=700*pow(daily,0.7);


/* Cost Ratios - Equivalent feet of 8x8-ft drift */

drift_8x8=0.0825*pow(drift_area,0.6);

ramp_8x8=0.0970*pow(ramp_area,0.6);

serv_8x8=0.0948*pow(serv_area,0.6);

/* Development Cost Calculation */
/* $148/ft of 8x8-ft drift - 1988 (base year) */

develop_cost= drift_8x8*drift_leng+ramp_8x8*ramp_leng+serv_8x8*serv_leng;

develop_cost= develop_cost*148;


/* Cost of Drilling, Loading, and Haulage Equipment - Modified*/

equip_cost=24600*pow(daily,0.8)/pow(stope_height,0.3);

/* Cost of Mine Ventilation */

fan_hp=0.88*pow(daily,0.9);

vent_cost=7500*pow(fan_hp,0.6);

/* Cost of Pumping System */
/* Little water inflow and Depth < 1000 ft */

pump_hp=8.0*pow(daily,0.5);

pump_cost=1400*pow(pump_hp,0.7);
```

```
/* Cost of Water System */

water_cost=5300*pow(daily,0.4);

/* Cost of Primary Crusher Ugd */

jaw_cost=1370*pow(daily,0.6);

jaw_inst=210*pow(daily,0.7);

/* Cost of Maintenance Shop */

shop_cost=14600*pow(daily,0.4);

/* Cost of Mine Compressor Plant */
/* stope-pillar: difficult air circulation */

air_need=230*pow(daily,0.5);

air_cost=920*pow(air_need,0.7);

/* Cost of Air & Water Distribution - Modified*/

lat_ext=1276*pow(daily,0.6)/pow(stope_height,0.4);

pipe_cost=2.80*pow(lat_ext,0.9)*pow(air_need,0.3);

/* Electrical Distribution */

elect_load=24.75*pow(daily,0.5);

elect_cost=1600*pow(elect_load,0.9);

/* Total Capital Cost */

ugd_dev= shaft_costfix + shaft_costdep + develop_cost + shop_cost;

ugd_dev= ugd_dev* capex_factor;

ugd_equip= hoist_cost + hoist_install + hoist_room + frame_cost + bin_cost;

ugd_equip= ugd_equip + equip_cost + vent_cost + pump_cost + water_cost;

ugd_equip= ugd_equip + jaw_cost + jaw_inst + air_cost + pipe_cost + elect_cost;

ugd_equip= ugd_equip* capex_factor;

ugd_capex = ugd_dev + ugd_equip;

/* Tax Credit Calculation */
/* Depreciation on Development @ 100% & Equipment DBD @ 30% */
/* Equipment Depreciation starts only after pre-production */
```

```
for (temp=3; temp<=LIFE; temp++)
{
        tax_cred[temp] = ugd_equip*pow((1-dbd_rate),(temp-3))*dbd_rate;
        tax_cred[temp]= tax_cred[temp]*tax_rate;
}

/* Development Depreciation splitted over 2 years */

tax_cred[1]= 0.5*ugd_dev*tax_rate;
tax_cred[2]= 0.5*ugd_dev*tax_rate;


/* Splits Capex over 2-year pre-production */


ug_flow[0]=0;

ug_flow[1]=capex_perc[1]*ugd_capex/100;

ug_flow[1]=(ug_flow[1]-tax_cred[1])/1000000;

ug_flow[2]=capex_perc[2]*ugd_capex/100;

ug_flow[2]=(ug_flow[2]-tax_cred[2])/1000000;

/* Daily Operating Underground Costs */

correc_fac= pow((12/stope_height),0.4);

/* Sublevel Stoping Method */

rp_opcost=160*pow(daily,0.6)*correc_fac;

/* Daily Crushing & Hoisting Costs */

crush_opcost=2*pow(daily,0.8);

hoist_opcost=4.70*pow(daily,0.8);


/* Annual Operating Costs */

ugd_opcost = (rp_opcost + crush_opcost + hoist_opcost)*days_year*op_factor;

for (time=3; time<=LIFE; time++)
{
        ug_flow[time]=ugd_opcost * otf - tax_cred[time];
        ug_flow[time]=ug_flow[time]/1000000;
}

/* Save Results in ASCII file*/
clrscr();
clear_kb();
puts ("Save Results in ASCII file.");
```

```
        puts ("Enter file name with extension.");
        gets (filename);

        if ((fp=fopen(filename,"w")) == NULL)

        {
                fprintf(stderr, "Error opening file %s.", filename);
                exit(1);
        }

        for (time=1; time<LIFE; time++)
        {
                fprintf (fp,"\n%d",time);
                fprintf (fp,"\t%.2f",op_flow[time]);
        }

        fclose(fp);

}

/* ============================================================*/
/* Present Value Calculations and Comparison */

/* ============================================================*/

void calc(float op_flow[], float ug_flow[])
{
        float op_pc[LIFE], ug_pc[LIFE], pre_cost;
        int time, ugd_life, life_left, ugd_ini;
        int brk_even, pre_prod;


        /*Call the Present Worth function for Open Pit*/

        for (time = 0; time<LIFE; time++)

        {
                op_pc[time]=prescost(op_flow,time,LIFE,disc_rate);
                printf("\nOP Present Worth for Time %d is %f",time,op_pc[time]);
        }

        /*Call the Present Worth function for Underground*/
        /* UG Pre-production period is 2 years : pre_cost */

        life_left=LIFE;

        ugd_life=life_left - min_life;

        ugd_ini=2;

        pre_cost = ug_flow[0]*pow((1+disc_rate),1)+ug_flow[1];

        for (time = 2; time<=ugd_life; time++)
```

```
        {
                ug_pc[time]=prescost(ug_flow,ugd_ini,life_left,disc_rate);

                ug_pc[time]=ug_pc[time]+pre_cost;

                printf("\nUG Present Worth for Time %d is %f",time,ug_pc[time]);
                life_left=life_left-1;

        }

        /* Compares OP & UG Present Worths and Displays the Result */
        /* UG pre-production is 2 years and minimum UG life is taken into account */

        for (time=2; time<=ugd_life; time++)

        {

                if (op_pc[time]>=ug_pc[time])
                {
                        brk_even=time;

                        printf("\nBreak-Even Time is %d",brk_even);

                        pre_prod=brk_even-2;

                        printf("\nPre-Production should start at Time %d",pre_prod);

                        break;
                }

                else
                        if (time==ugd_life)
                                printf("\nThere is no Break-Even Time");

    }


}
```

```
/* Function Present Worth */

float prescost(float x[], int y, int z, float k)
{

        int time, n_years;
        float result;

        result=0;
        time=0;

        k=1+k;

        for( time=y; time<z; time++)

        {
                n_years= time-y+1;

                result=result + x[time]/pow(k,n_years);

        }

        return result;

}
```

IMAGE EVALUATION TEST TARGET (QA-3)

150mm

6"