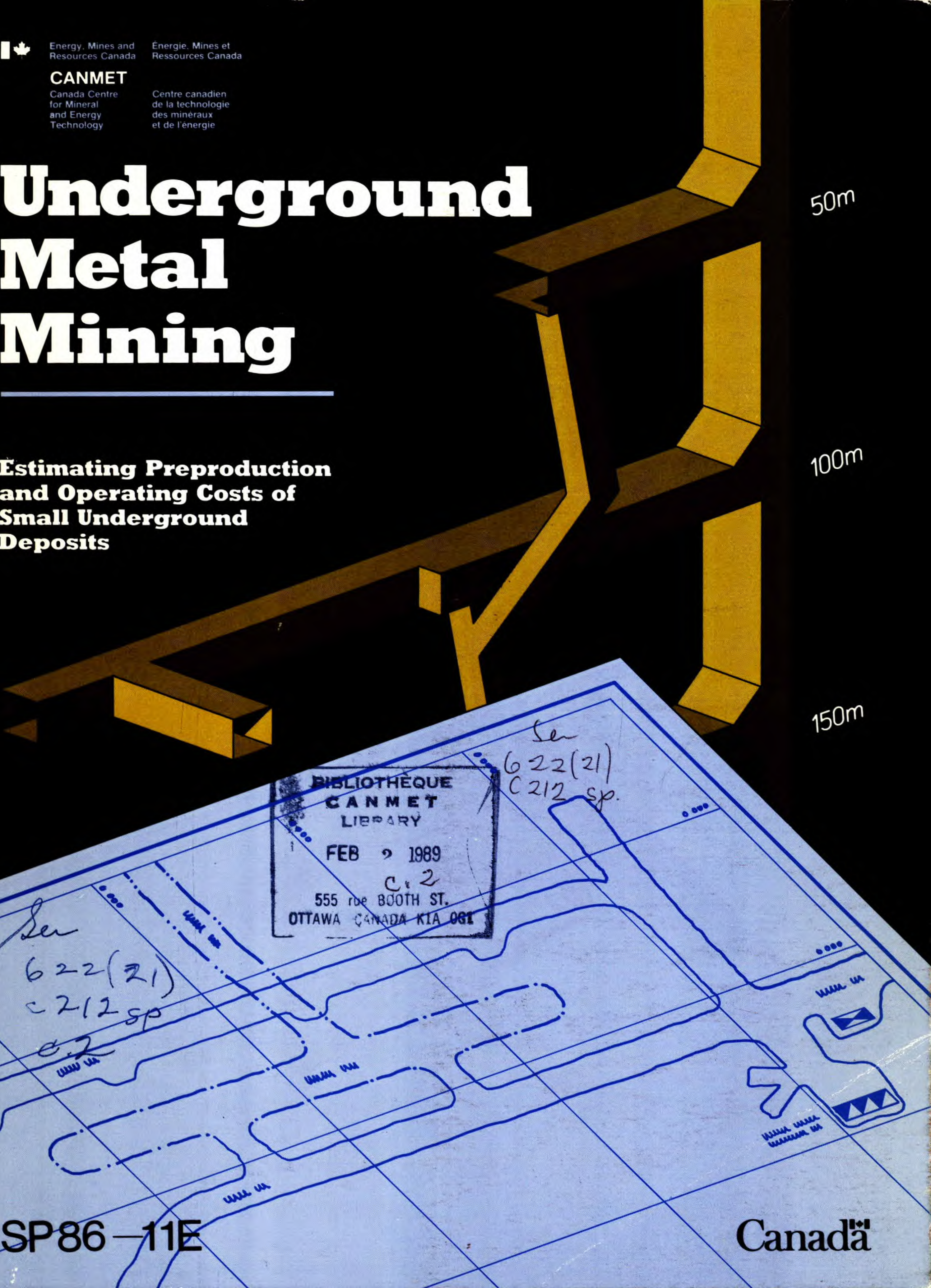
Energy, Mines and Resources Canada
Énergie, Mines et Ressources Canada

CANMET
Canada Centre for Mineral and Energy Technology
Centre canadien de la technologie des minéraux et de l'énergie

Underground Metal Mining

Estimating Preproduction and Operating Costs of Small Underground Deposits



BIBLIOTHÈQUE
CANMET
LIBRARY
FEB 2 1989
C. 2
555 rue BOOTH ST.
OTTAWA CANADA K1A 0G1

Ser
622(21)
C212 sp.

Ser
622(21)
C212 sp
c.2

SP86—11E

Canada

ESTIMATING PREPRODUCTION AND OPERATING COSTS OF SMALL UNDERGROUND DEPOSITS

by

J.S. Redpath Limited
710 McKeown Avenue
North Bay, Ontario
P1B 8K1

Work on this project was conducted under the auspices of
CANMET
Energy, Mines and Resources Canada.

DSS FILE NO. :092Q.23440-5-9128
DSS CONTRACT SERIAL NO. :1SQ85-00199
CANMET PROJECT NO. :140901
SCIENTIFIC AUTHORITY :R.W.D. CLARKE P.Eng.

November 1986

© Minister of Supply and Services Canada 1987

Available in Canada through

Associated Bookstores
and other booksellers

or by mail from

Canadian Government Publishing Centre
Supply and Services Canada
Ottawa, Canada K1A 0S9

Catalogue No. M38-15/86-11E
ISBN 0-660-12247-2

Price subject to change without notice

Disponible en français

Reprinted 1988

FOREWORD

Small mines, defined as those with productions of less than 500 tonnes per day, are a vital component of the Canadian mining industry. Often, they are the result of exploration activities carried out by junior mining companies, prospectors, developers, and syndicates of independent entrepreneurs with limited financial resources. Recognizing that the principal vocation of many of those groups is the discovery of new deposits, it was considered of value to them, and to the Canadian economy, if these costs associated with the carrying out of preliminary assessments of the economic viability of promising finds could be minimized.

With this objective in mind, CANMET awarded a contract to J.S. Redpath Ltd., to develop a manual in respect of the estimation of pre-production and operating costs for small underground deposits. The cost information and preliminary engineering analytical procedures contained in this manual are based on their wide experience. To ensure that it meets industry requirements, the manual was reviewed regularly during its preparation by experienced members of the mining industry. Their comments and suggestions have been incorporated as much as possible into the final text.

The manual is not intended to replace professional feasibility studies of mining properties. Rather, it is to assist the finders of deposits in their decisions to proceed to such studies. We believe that the manual prepared by J.S. Redpath Ltd., admirably meets this objective.

John E. Udd
Director
Mining Research Laboratories

TABLE OF CONTENTS

NOTE: A detailed Table of Contents is presented at the beginning of each section.

1.0 INTRODUCTION

SECTION 1

1.1 INTRODUCTION - PURPOSE AND LIMITATIONS

The object of this manual is to provide a step-by-step approach to estimating capital and operating costs for small underground mines producing 500 tonnes per day or less.

The manual is applicable only to hard rock mines within Canada. Minerals that require special consideration such as potash, or coal are not addressed. The manual concentrates on those costs associated with mine development and mining. The capital and operating costs of milling are included but these items are dealt with in a more general manner.

The information presented in this manual is based largely on J. S. Redpath Limited's extensive operating experience in every Canadian province and territory. Much of the cost information was developed from first principles and verified against actual project records. Some cost data was also obtained from active Canadian mining operations.

The simplified cost estimating procedures used in the manual are not meant to replace a professional feasibility study. They will, however, provide a preliminary indication regarding the viability of a small mineral deposit. The accuracy of the cost estimates developed by the user is expected to be in the range of plus or minus 30%. The accuracy will be significantly influenced by the quality of information provided by a user.

While the main purpose of the manual is to provide cost information, sections 5.0 and 6.0 assist the user in estimating the value of the mineral deposit thus leading to a preliminary indication of economic viability.

The user should now read Sections 1.2, 1.3 and 1.4 which explain the use of the manual and describe the forms. The remainder of the manual can then be used as a reference document to assist in developing estimates of costs.

Note:

It is assumed that the user comes to the manual with information regarding tonnes, grade and ore definition and requires capital and operating costs in order to make a future production decision. Those who do not have this information should establish the 'geological tonnes and grade' by completing Forms 5 a) and 5 b). Guidelines for the completion of these forms can be found in Section 5.2. This information is required before starting Section 2.0.

1.2 HOW TO USE THIS MANUAL

There are six sections in the body of the manual:

2.0	Operating Costs
3.0	Capital Costs
4.0	Regional Cost Factors
5.0	Mineral Deposit Value
6.0	Preliminary Cash Flow Summary
7.0	Exploration Programmes

The four basic steps to producing the cost estimates are outlined on page 1 - 5.

Sections 2.0 to 5.0 cover the two basic elements required for the economic analysis of any project ie. costs and revenues. The preliminary cash flow summary is developed in Section 6.0.

The order in which information is presented follows the decision making process as opposed to the chronological order in which work would be carried out. Many decisions made regarding the operation of a mine will influence the design of capitalized mine plant and excavations and, for this reason, operating costs are presented first.

The user may, from time to time, deviate from the sequence presented in the manual. Decisions relating to operating and capital costs are inter-related and some movement between Sections 2.0 and 3.0 may be required to optimize costs. All of the Sections, 2.0 to 5.0 however, must be completed to assess the viability of a project.

The user may not have all the information required to carry out this analysis and may wish to use the manual simply to develop preproduction capital costs for an exploration project. The manual has been formatted to accommodate this need. Section 7.0 outlines how to develop capital costs for an underground exploration programme.

Worked examples and calculation forms are included in the appendices. These lead the user through the steps required to complete an economic analysis of a project.

In general, estimates are completed using "fill-in-the-blank" type forms found in Appendix B.

The forms, numbered 1 to 6, should be completed in that order as estimates sometimes refer to a previous calculation or decision. Sections 2.0 to 6.0 contain guidelines and cost information to assist in filling in the forms.

Most information is presented in either graphical or tabular form. The user is often required to make a selection from a range of costs or to adjust a calculation to suit the specifics of a particular site.

In each subsection, cost information is preceded by a description of the items included. The manual attempts to include those items 'typical' of small underground mines but naturally, specific requirements will vary. The user must evaluate the content and, if necessary, adjust the costs presented.

The cost information has been compiled based on the conditions prevailing in north-central Ontario. To account for the variations in costs from one area to another, regional cost factors have been developed and are presented in Section 4.0.

Several worked examples of the complete estimating process are included in Appendix A for clarification.

1.3 FOUR BASIC STEPS TO PRODUCING THE COST ESTIMATES

STEP ONE

It is assumed that the user has some basic knowledge of the property.

This should include:

- ° location
- ° local surface conditions
- ° approximate size, shape and depth of ore body
- ° rock and ground water conditions

List this information on Form 1 "Basic Information".

STEP TWO

With the basic information from STEP ONE, other key decisions must be made.

These decisions include:

- ° production rate
- ° means of mine access
- ° mining method
- ° source of milling

(Help in making these decisions is provided in Section 2.0. Enter these decisions in the appropriate space on Form 2a).

STEP THREE

The user can now complete the estimates of capital & operating costs.

Operating costs should be entered on Form 2a) using information contained in Section 2.0.

Capital costs should be entered on Forms 3a) and 3b) using information contained in Section 3.0.

STEP FOUR

Totals of capital & operating costs are then multiplied by a regional cost factor to allow for cost differences between different parts of Canada.

The cost factors are found and explained in Section 4.0.

1.4 DESCRIPTION OF THE CALCULATION FORMS

Each form is briefly described below:

FORM ONE	BASIC INFORMATION Summarizes information concerning project location, site description and geological conditions.
FORM TWO	OPERATING COSTS Develops operating costs on a step-by-step basis. Key decisions regarding mine access, mining method, etc. are made.
FORM THREE	CAPITAL COSTS Develops capital costs in two sections: a) Preproduction capital costs; b) Ongoing capital costs.
FORM FOUR	REGIONAL COST FACTORS Develops factors for both operating and capital costs to adjust estimates for site location.
FORM FIVE	MINERAL DEPOSIT VALUE Outlines calculations of geological and mineable tonnages and grade. Recovery and dilution factors are then applied and by estimating the product selling price, a value per tonne milles is obtained.
FORM SIX	PRELIMINARY CASH FLOW SUMMARY Compares costs to revenues to arrive at a preliminary indication of the project's financial viability.
NOTE:	The form numbers relate directly to the sections containing the appropriate information.

SECTION 2

2.0 OPERATING COSTS

2.1 INTRODUCTION AND CRITERIA

General

Operating costs and manpower loadings compiled using information contained in this section should be entered and summarized on Forms 2(a) and 2(b).

The costs presented in this section are based on specific layouts and parameters, descriptions of which accompany the cost information. While the parameters and costs presented are "typical" of small mines, each property is unique and costs will vary accordingly. The user must compare the parameters described in the manual to those anticipated for the property being evaluated, and adjust cost information accordingly.

Cost Criteria

The following criteria have been used in developing Operating Costs:

1) Costs are in first quarter 1986 Canadian dollars.

2) Operating costs are presented for a base case property located in north-central Ontario. Operating costs selected by the user must be adjusted to suit the actual geographical location of his property through the use of Regional Cost Factors found in Section 4.0.

3) All units are metric.

4) Costs assume road access is available.

2.2 SELECTION OF PRODUCTION RATE

There are a number of formulae available in the industry for determining the optimum mine production rate. None are perfect for all conditions. The formula presented here is "Taylor's Rule" for determining mine life. It is purely an empirical relationship based on Mr. Taylor's work records and experience accumulated during his lengthy career in mine evaluation, planning and operation. It claims no more than to make a preliminary selection of the range within which an economic and attainable rate is likely to lie, and it is not a substitute for detailed study.

$$\text{Mine Life (years)} \quad = \quad 0.20\ (\text{expected ore tonnes})^{0.25}$$

In order to determine production rate, the following formula is derived from Taylor's Rule.

$$\text{Production rate (tonnes/day)} = \frac{5(\text{expected ore tonnes})^{0.75}}{\text{working days per year}}$$

"Expected ore tonnes" should be interpreted broadly as a reasonable expectation of mineable ore which may be somewhat greater than a declared and proved ore reserve.

		Daily Production Rate		
Expected Ore Tonnes	Mine Life in Years	250 Working Days /Year	300 Working Days /Year	350 Working Days /Year
100,000	3.6	112	94	80
150,000	3.9	152	127	109
200,000	4.2	189	158	135
250,000	4.5	224	186	160
300,000	4.7	256	214	183
350,000	4.9	288	240	206
400,000	5.0	318	265	227
450,000	5.2	347	290	248
500,000	5.3	376	313	269
600,000	5.6	431	359	308
700,000	5.8	484	403	346
800,000	6.0	535	446	382
900,000	6.2	584	487	417
1,000,000	6.3	632	527	452

2.3 SELECTION OF MINING METHOD

Four basic mining methods, suitable for mining small ore bodies in the 100 to 500 tpd production range, are described in this section. The selection of the mining method is largely dependent on the following criteria:

1. Geometry of ore body (dip, thickness, etc.)
2. Rock competence - nature of ore, hanging wall and footwall rocks.
3. Definition and continuity of ore body.
4. Production rate required.
5. Recovery and dilution considerations.

Consideration should be given to all of the above factors as well as any site specific considerations that may need to be addressed.

The four mining methods are described on the following pages. Each includes an outline as well as a list of advantages, disadvantages and essential criteria where applicable. The user should review each mining method as presented and select the mining method(s) best suited to his mineral deposit.

2.3.1 Blasthole Stoping

General

Blasthole stoping is an open stoping method that is normally confined to fairly regular ore bodies where both the ore and country rock require little support during mining activities. The method is characterized by a comparatively high development to stoping ratio, which is compensated for somewhat by the fact that the majority of development is in ore. (See Fig. 1)

Essential Criteria

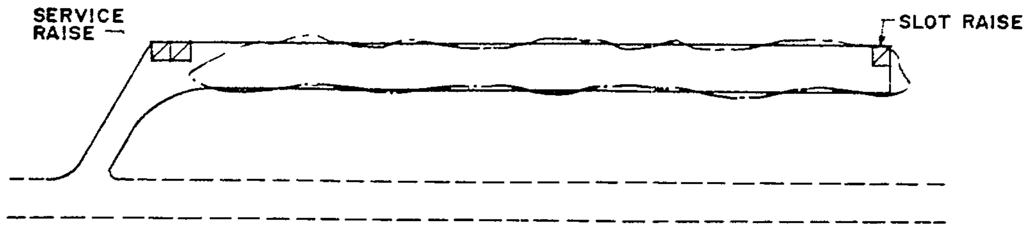
- Dip must be greater than the angle of repose of broken ore or 50°.
- Thickness must be greater than 3 m.
- Ore and country rock must be competent with well defined footwall and hanging wall contacts.

Advantages

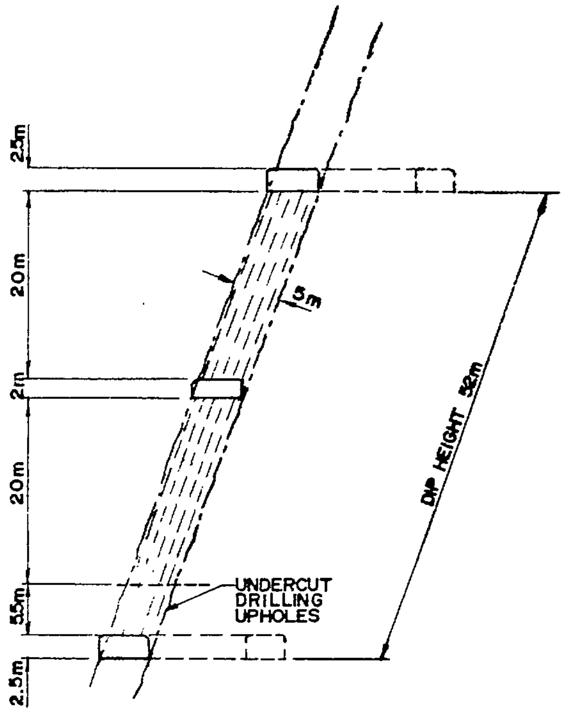
- Good recovery, moderate dilution
- Excellent productivity
- Safe method
- Low cost/tonne method
- Good ventilation
- Amenable to mechanization
- Moderately flexible method.

Disadvantages

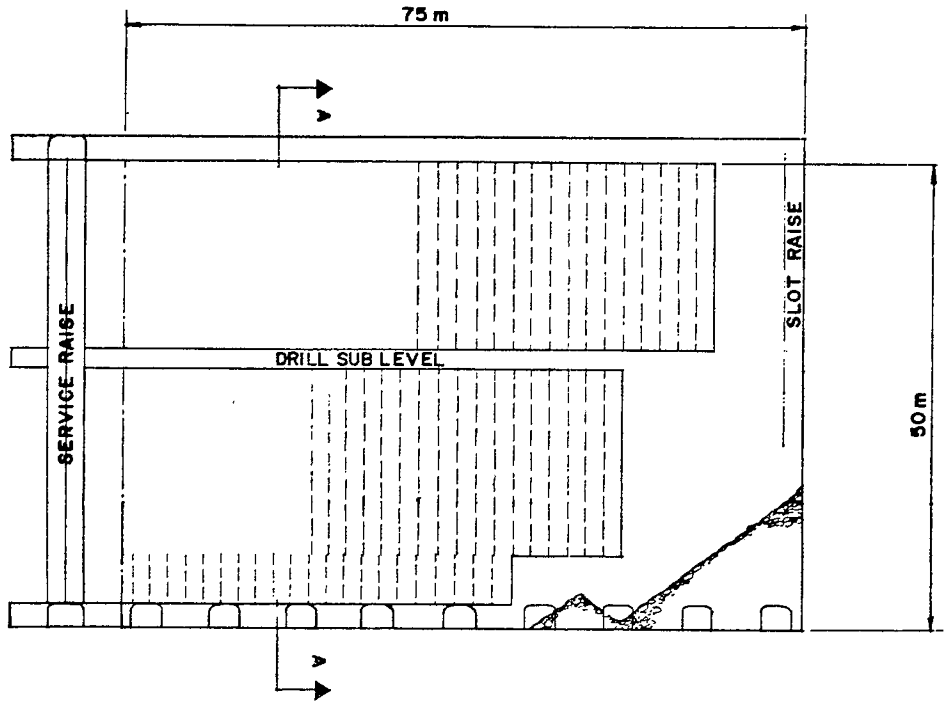
- Considerable preproduction development
- Secondary blasting costs can be high
- Poor recovery of ore shoots
- In poor rock conditions dilution will be high.



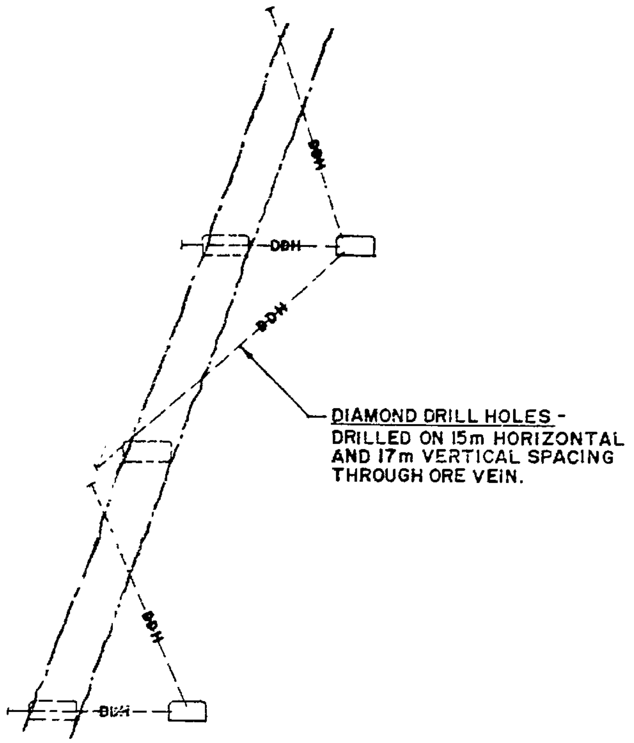
UPPER LEVEL PLAN



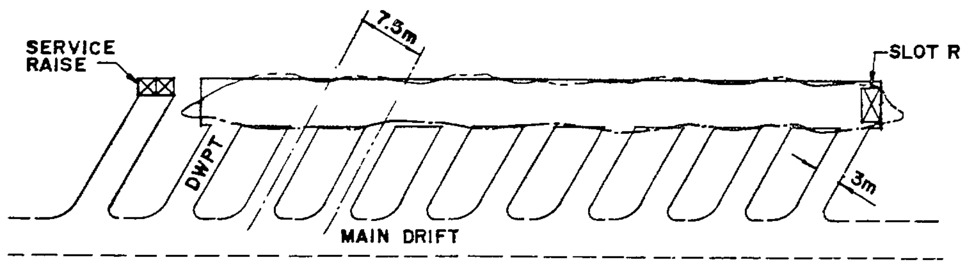
CROSS-SECTION A-A
PRODUCTION DRILLING



BLASTHOLE STOPING
LONGITUDINAL SECTION



CROSS-SECTION A-A
DIAMOND DRILL HOLES



LOWER LEVEL PLAN

		5							2						
		4							1						
		3							0	FIRST DRAFT	APR.86	WJB			
DWG NO.	REFERENCE DRAWING	NO.	REVISIONS	DATE	BY	CHECK	CHIEF ENG.	PROJ. ENG.	NO.	REVISIONS	DATE	BY	CHECK	CHIEF ENG.	PROJ. ENG.

J S REDPATH · LIMITED
MINING CONTRACTORS and CONSULTING ENGINEERS

CLIENT D.S.S.

TITLE SCHEMATIC
BLASTHOLE STOPING

DATE APRIL'86 | SCALE 1:400 | DWG NO. FIG. 1 | REV 0

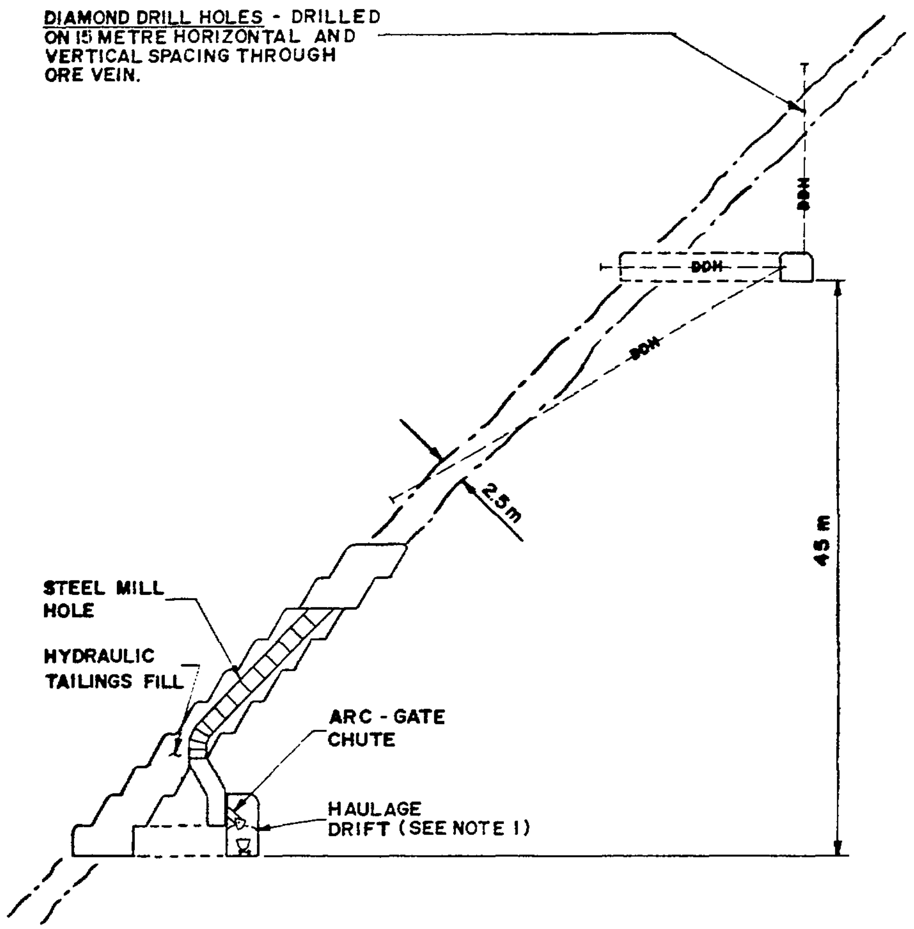
CLIENT D.S.S.
TITLE SCHEMATIC - BLASTHOLE STOPING
DWG. NO. FIG. 1
REV 0

REV. 0

DWG. NO. FIG. 2

TITLE SCHEMATIC - CUT & FILL STOPING

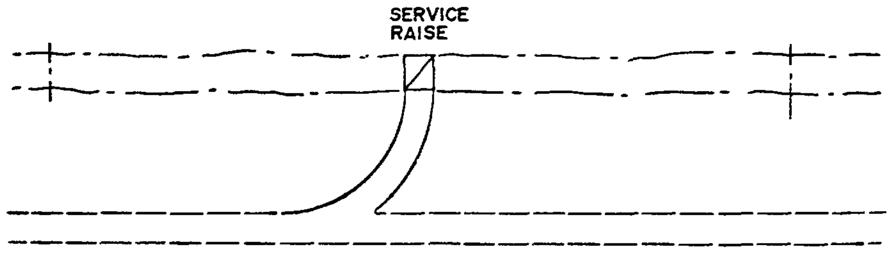
CLIENT D.S.S.



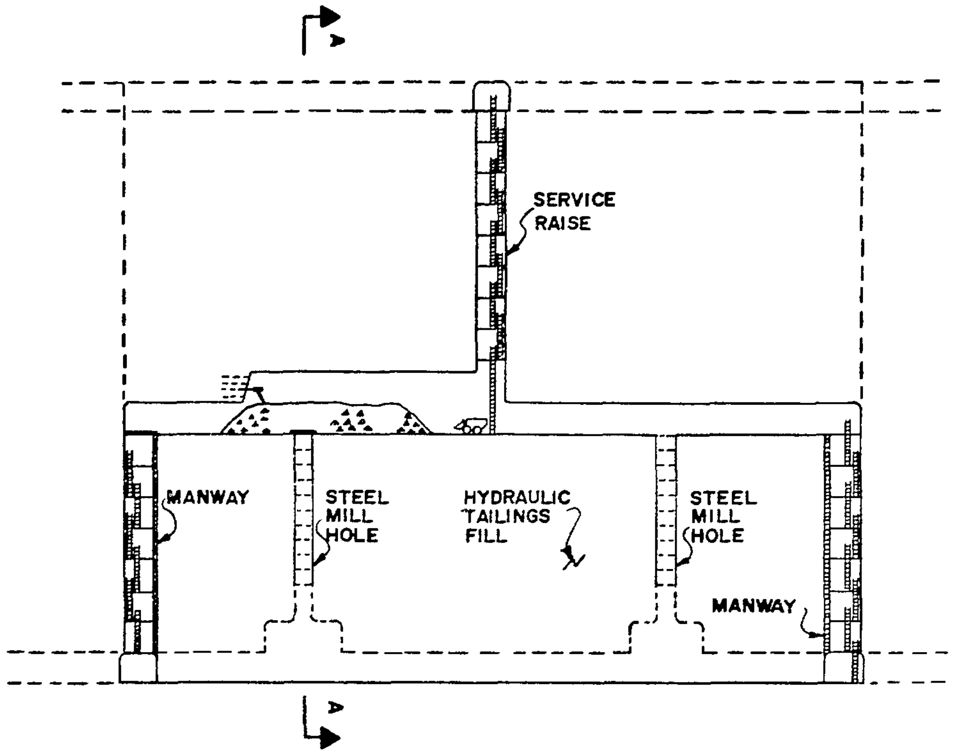
CUT - AND - FILL STOPING

CROSS SECTION A-A

NOTE 1: HAULAGE DRIFT MAY BE LOCATED IN ORE. BACK OF DRIFT WOULD BE TIMBER SUPPORTED TO HOLD FILL.

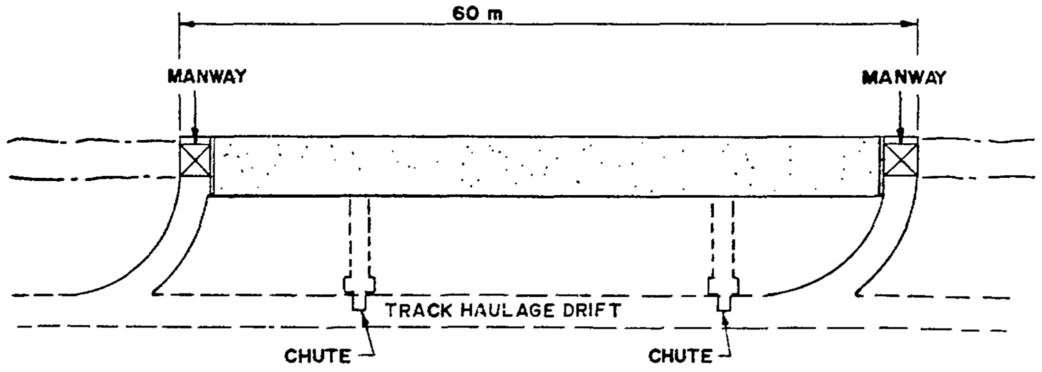


UPPER LEVEL - PLAN



CUT - AND - FILL STOPING

LONGITUDINAL SECTION



LOWER LEVEL - PLAN

JS REDPATH·LIMITED

MINING CONTRACTORS and CONSULTING ENGINEERS

CLIENT D.S.S.

TITLE
SCHEMATIC
CUT & FILL STOPING

DATE	SCALE	DWG NO	REV.
APRIL 86	1:300	FIG. 2	0

DWG NO.	REFERENCE DRAWING	NO.	REVISIONS	DATE	BY	CHECK	CHIEF ENG.	PROJ. ENG.	NO.	REVISIONS	DATE	BY	CHECK	CHIEF ENG.	PROJ. ENG.
		5							2						
		4							1						
		3							0	FIRST DRAFT	[illegible]86				

2.3.2 Cut and Fill Stoping

General

The cut and fill mining method extracts small horizontal slices of ore, which are partially or completely filled before the next slice is extracted. Supported openings are extended through the fill for access, ventilation, drainage and ore removal. The most common application for filled stopes is in ore bodies with a moderate to steep dip, with restrictive dimensions and weak walls, where high recovery or selective mining is desirable. (See Fig. 2).

Essential Criteria

- There must be an available source of fill.
- Dip must be at least 40°.
- Steeper dip is required for very narrow veins.
- Grade must be above average because mining costs are high.

Advantages

- High recovery, low dilution
- Waste can be left in stope as fill
- Good regional ground support, subsidence inhibited
- Good ventilation
- Relatively safe method
- Very flexible and selective method
- Amenable to mechanization
- Easy mining of parallel veins and shoots

Disadvantages

- Costs of using fill
- Filling cycle slows down mining
- Must mine from the bottom of the ore body up to avoid sill pillars
- Low productivity

2.3.3 Shrinkage Stoping

General

Shrinkage methods are most often used in steeply dipping vein-type deposits, where the ore and walls are competent enough to stand with little or no support. Slabbing and minor weakness in the country rock can be tolerated as long as resulting dilution is not a serious problem, but severe slabbing will plug the chutes and squeezing will bind the broken ore in place. The ore must be competent because it is usually uneconomical to provide anything but localized support to the back. (See Fig. 3).

Essential Criteria

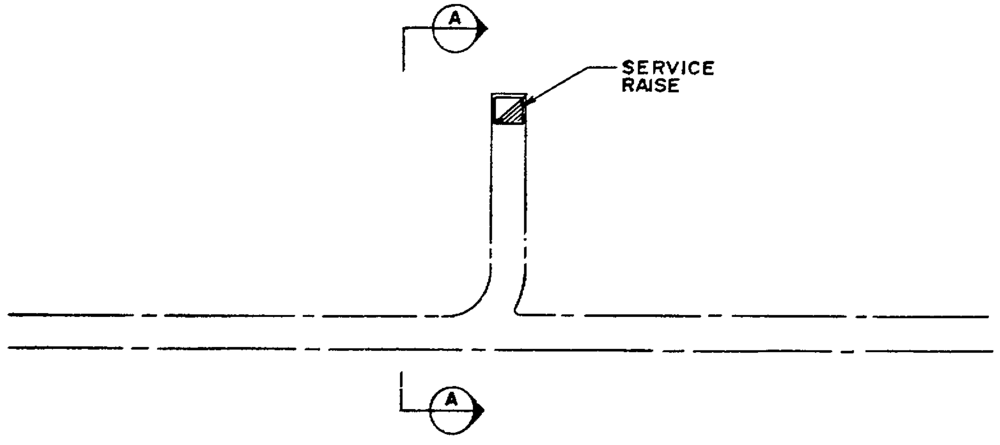
- Dip must be greater than the angle of repose of broken ore or 50°.
- Must have competent ore and wall rock.

Advantages

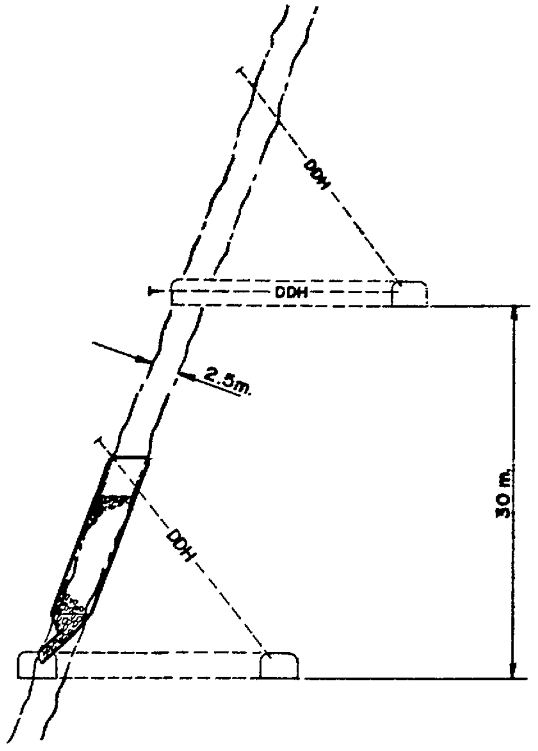
- High recovery, low dilution
- Relatively safe method
- Good ventilation
- No backfill required
- Stope equipment costs low
- Flexible method
- Easy mining of ore shoots

Disadvantages

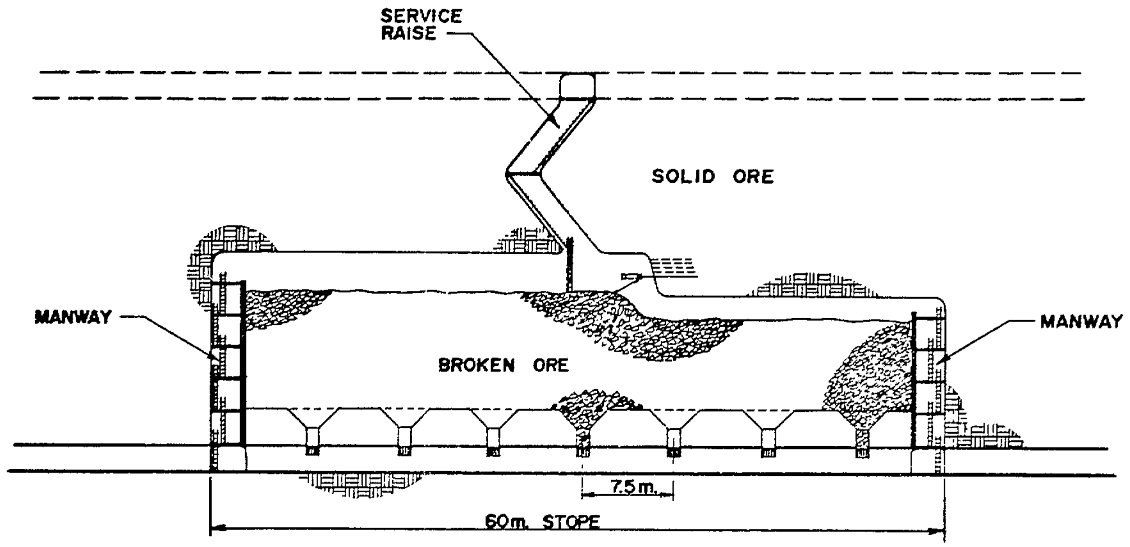
- Ore tied-up in producing stopes
- Whilst this method gives excellent ground control during mining, it can lead to excessive dilution in poor ground when drawing stope empty.



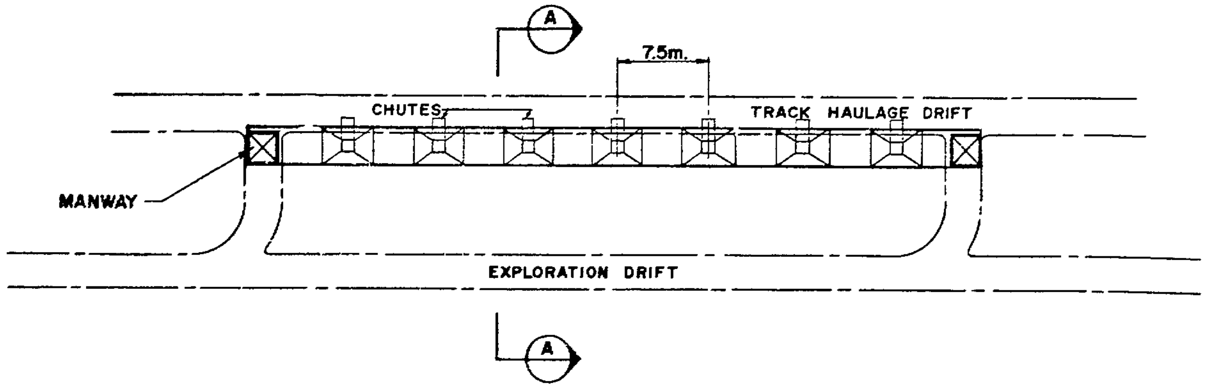
UPPER LEVEL PLAN



CROSS-SECTION (A-A)



LONGITUDINAL SECTION



LOWER LEVEL PLAN

DWG NO.	REFERENCE DRAWING	NO.	REVISIONS	DATE	BY	CHECK	CHIEF ENG.	PROJ. ENG.
		5						
		4						
		3						

NO.	REVISIONS	DATE	BY	CHECK	CHIEF ENG.	PROJ. ENG.
2						
1						
0	FIRST DRAFT	APR.'86				

JS REDPATH·LIMITED
MINING CONTRACTORS and CONSULTING ENGINEERS

CLIENT D.S.S.

TITLE SCHEMATIC
SHRINKAGE STOPING

DATE	SCALE	DWG NO.	REV.
APR.'86	1:300	FIG. 3	0

CLIENT D.S.S.
TITLE SCHEMATIC SHRINKAGE STOPING
DWG. NO. FIG. 3
REV 0

2.3.4 Room and Pillar Stoping

General

Room and pillar mining derives its name from the basic approach: driving openings in the ore and leaving pillars to provide support for the hanging wall rocks. There are many variations to the method to account for individual ore body characteristics. Room and pillar mining is normally used in ore bodies with a dip of less than 40°, and with moderate to large lateral extent. The size of the pillars and the rooms is primarily dependent on ground conditions and the thickness of the deposit. Flat deposits, especially the thicker ones, can be highly mechanized resulting in high productivity rates and low mining costs. Such deposits typically result in larger mines with production rates above 500 tpd. The example depicted in Figure 4 is more common to small mines. Here there is very little mechanization, thus productivities are low and costs relatively high.

Essential Criteria

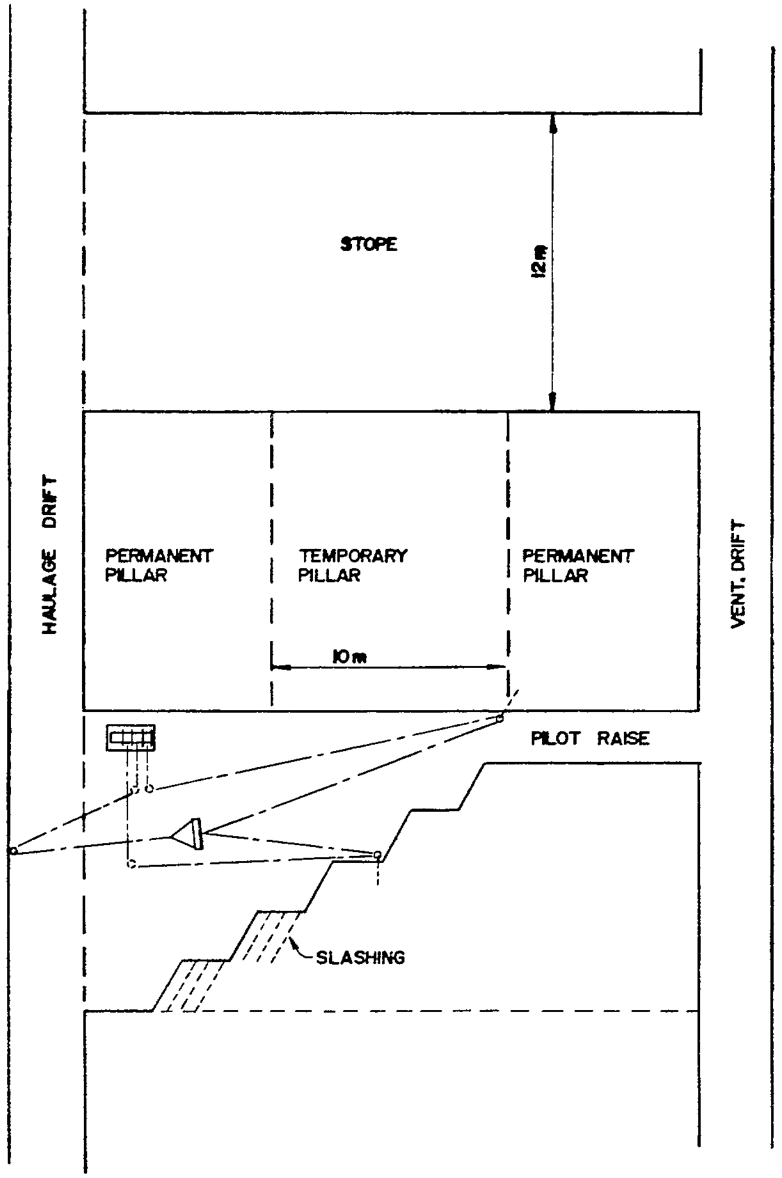
- Dip must be less than 40°.

Advantages

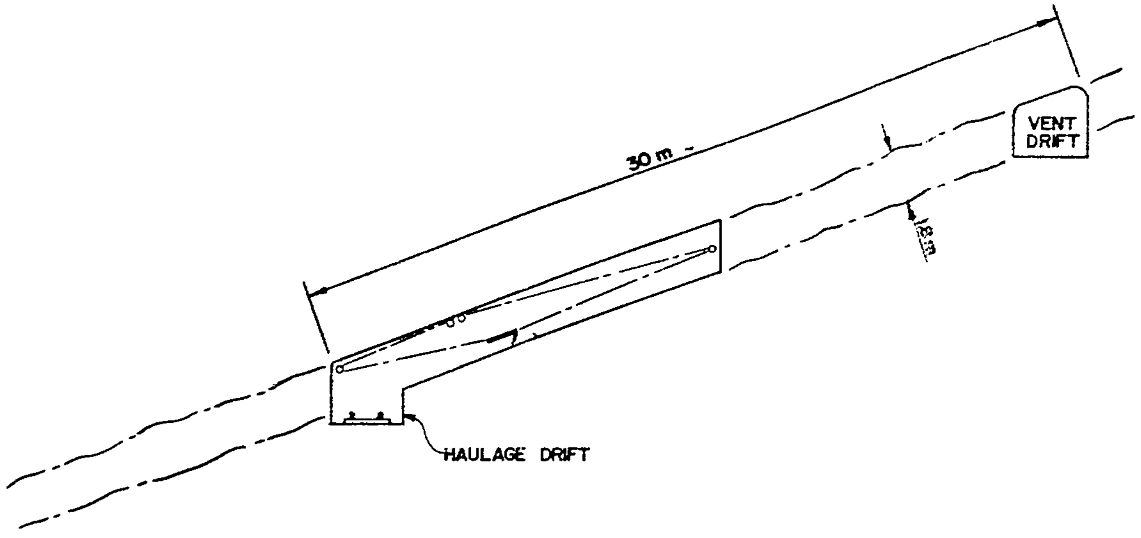
- Low dilution
- High degree of selectivity possible
- Relatively flexible method
- Good regional ground support, subsidence inhibited
- Moderately good ventilation
- Relatively safe method
- Amenable to mechanization
- Frequently low preproduction development requirements

Disadvantages

- Moderate recovery (loss of recovery in pillars)
- Ground support costs may be high
- Ventilation costs high
- Productivity is quite low if mechanization is not practical.



ROOM AND PILLAR
STOPING - PLAN



ROOM AND PILLAR
STOPING - SECTION

REV. 0
DWG. NO. FIG. 4
TITLE SCHEMATIC - ROOM AND PILLAR STOPING
CLIENT D.S.S.

DWG NO.	REFERENCE DRAWING	NO	REVISIONS	DATE	BY	CHECK	CHIEF ENG.	PROJ. ENG.	NO.	REVISIONS	DATE	BY	CHECK	CHIEF ENG.	PROJ. ENG.
		5							2						
		4							1						
		3							0	FIRST DRAFT	JUNE '66	E.			

JS REDPATH · LIMITED
MINING CONTRACTORS and CONSULTING ENGINEERS

CLIENT
D.S.S.

TITLE
SCHEMATIC
ROOM AND PILLAR STOPING

DATE	SCALE	DWG NO.	REV.
JUNE '66	—	FIG. 4	0

2.4 STOPING COSTS

2.4.1 Blasthole Stoping

2.4.1 a) Description

Figure 1 shows a typical blasthole stoping layout showing drill sublevels, drawpoint production level and service and slot raises.

Stoping costs include definition diamond drilling, development, mining and drawpoint mucking. Drawpoint mucking in this case is scooptram mucking from the drawpoint to the ore pass. Haulage beyond this point is level haulage and is identified as a separate cost (see Section 2.7).

The user of the manual should be aware that the particular deposit under evaluation may vary somewhat from the typical layout shown here, therefore, a range of overall stoping costs is offered with the example suitably positioned within the range. The user should compare the deposit being evaluated with the layout and data shown here and select a stoping cost accordingly.

The data used to develop stoping costs is listed below:

Stope dimensions:	
Strike length (m)	75
Thickness (m)	5
Dip height (m)	52
Volume (m^3)	19,500
Tonnage factor (tonnes/m^3)	3.0
Stope tonnage (tonnes)	58,500
% Ore from stoping	89%
% Ore from development	11%
Labour productivity (tonnes/m.s.)	47
Haul distance from drawpoint to ore pass (metres)	150

The major factors that would increase or decrease stoping costs are listed below:

Factors Increasing Costs	Factors Decreasing /Costs
Ore density less than 3.0 tonnes/m³	Ore thickness greater than 5 m
Increased development/ tonne mined	Ore density greater than 3.0 tonnes/m³
Adverse ground conditions	Decreased haul distance to ore pass or main haulage drift
Substantial water inflows	

2.4.1 b) Blasthole Stoping Costs

	ITEM	$/TONNE
1.	Diamond Drilling	$ 0.34
2.	Stope Development	4.33
3.	Stoping Labour	1.36
4.	Drawpoint Mucking	1.31
5.	Drilling Supplies - Bits & Steel, etc.	1.11
6.	Blasting Supplies - Powder & Accessories	
	- Primary	0.58
	- Secondary	0.06
7.	Ground Support (included in item 2)	-
8.	Pipe, Timber, Aux. Ventilation & Misc. Supplies	0.12
9.	Equipment Operating and Maintenance Supplies	0.89
	Subtotal	$10.10
	Misc. Costs @ 10%	1.01
	TOTAL	$11.11

Blasthole Stoping Cost Range $9.00 - $15.00

2.4.2 Cut and Fill Stoping

2.4.2 a) Description

Figure 2 shows a typical cut and fill stoping layout showing manway and service raises, mill holes to haulage level, backfill emplaced, breast mining and Cavo mucking machine mucking to the mill holes.

Stoping costs include definition diamond drilling, development, mining, mucking to mill hole and backfill emplacement.

As shown in Figure 2, the broken ore is pulled from chutes directly into rail cars on the main haulage level and trammed to the shaft. This level haulage cost is identified in Section 2.7.

The user of the manual should be aware that the particular deposit under evaluation may vary somewhat from the typical layout shown here, therefore, a range of overall stoping costs is offered with the example suitably positioned within the range. The user should compare the deposit being evaluated with the layout and data shown here and select a stoping cost accordingly.

The data used to develop stoping costs is listed below:

Stope dimensions:	
Strike length (m)	60
Thickness (m)	2.5
Dip height (m)	60
Volume (m^3)	9,000
Tonnage factor (tonnes/m^3)	3.0
Stope tonnage (tonnes)	27,000
% Ore from stoping	93%
% Ore from development	7%
Labour productivity (tonnes/m.s.)	17

The major factors that would increase or decrease stoping costs are listed below:

Factors Increasing Costs	Factors Decreasing Costs
Adverse ground conditions	Ore thickness greater than 2.5m
Ore density less than 3.0 tonnes/m³	Ore density greater than 3.0 tonnes/m³
Substantial water inflows	
Use of coarse backfill which requires placement	
Increased use of cement in backfill for extra support	
Increased development/ tonne mined	

2.4.2 b) Cut and Fill Stoping Costs

	ITEM	$/TONNE
1.	Diamond Drilling	$ 0.75
2.	Stope Development	3.82
3.	Stoping Labour (includes stope mucking)	10.77
4.	Drilling Supplies - Bits & Steel, etc.	1.66
5.	Blasting Supplies - Powder & Accessories	1.40
6.	Ground Support Supplies	0.22
7.	Pipe, Timber, Aux. Ventilation & Misc. Supplies	1.49
8.	Equipment Operating and Maintenance Supplies	0.51
9.	Sandfill	4.25
	Subtotal	$24.87
	Misc. Costs @ 10%	2.49
	TOTAL	$27.36

Cut & Fill Stoping Cost Range	$22.00 - $32.00

2.4.3 Shrinkage Stoping

2.4.3 a) Description

Figure 3 shows a typical shrinkage stoping layout showing the drawpoint production level, breast mining and the service and ventilation development raises.

Stoping costs include definition diamond drilling, development, mining and drawdown of swell.

The broken ore is drawn from the stope via a boxhole chute arrangement directly into rail cars on the main haulage level. The level haulage costs are identified in section 2.7.

The user of the manual should be aware that the particular deposit under evaluation may vary somewhat from the typical layout shown here, therefore, a range of overall stoping costs is offered with the example suitably positioned within the range. The user should compare the deposit being evaluated with the layout and data shown here and select a stoping cost accordingly.

The data used to develop stoping costs is listed below:

Stope dimensions:	
Strike length (m)	60
Thickness (m)	2.5
Dip height (m)	32
Volume (m^3)	4,800
Tonnage factor (tonnes/m^3)	3.0
Stope tonnage (tonnes)	14,400
% Ore from stoping	82%
% Ore from development	18%
Labour productivity (tonnes/m.s.)	21

The major factors that would increase or decrease stoping costs are listed below:

Factors Increasing Costs	Factors Decreasing Costs
Adverse ground conditions requiring support of back and/or hanging wall or footwall drawpoints	Ore thickness greater than 2.5m
Reduced stope widths less than 2.5m	Ore density greater than 3.0 tonnes/m^3
Substantial water inflows	
Ore density less than 3.0 tonnes/m^3	
Increased development/ tonne mined	

2.4.3 b) Shrinkage Stoping Costs

	ITEM	$/TONNE
1.	Diamond Drilling	$ 0.88
2.	Stope Development	9.73
3.	Stoping Labour	5.01
4.	Drilling Supplies - Bits & Steel	1.47
5.	Blasting Supplies - Powder & Accessories	1.23
6.	Ground Support Supplies	0.20
7.	Pipe, Timber, Aux. Ventilation & Misc. Supplies	0.78
8.	Equipment Operating and Maintenance Supplies	0.05
	Subtotal	$19.35
	Misc. Costs @ 10%	1.94
	TOTAL	$21.29

Shrinkage Stoping Cost Range $18.00 - $27.00

2.4.4 Room and Pillar Stoping

2.4.4 a) Description

Figure 4 shows a typical slusher mining room and pillar layout for a thin vein, medium dipping ore body showing stopes and pillars, pilot raise development, slashing and slusher arrangement for slushing to main haulage drift.

Stoping costs include stope development, mining and slushing into ore cars.

The user of the manual should be aware that the particular deposit under evaluation may vary somewhat from the typical layout shown here, therefore, a range of overall stoping costs is offered with the example suitably positioned within the range. The user should compare the deposit being evaluated with the layout and data shown here and select a stoping cost accordingly.

The data used to develop stoping costs is listed below:

Stope dimensions:	
Dip length (m)	30
Strike width (m)	12
Thickness (m)	1.8
Volume (m^3) [includes temporary pillar & dev't.]	983
Tonnage factor (tonnes/m^3)	3.0
Stope tonnage (tonnes)	2949
% Ore from stoping	66%
% Ore from development	34%
Labour productivity (tonnes/m.s.)	21

The major factors that would increase or decrease stoping costs are listed below:

Factors Increasing Costs	Factors Decreasing Costs
Adverse ground conditions	Ore thickness greater than 1.8m
Substantial water inflows	Ore density greater than 3.0 tonnes/m^3
Ore density less than 3.0 tonnes/m^3	Good ground conditions permitting additional pillar recovery
Increased development/ tonne mined	Mechanized mining

2.4.4 b) Room and Pillar Stoping Costs

	ITEM	$/TONNE
1.	Diamond Drilling (from surface)	$ -
2.	Stope Development	8.57
3.	Stoping Labour (includes stope mucking)	5.37
4.	Drilling Supplies - Bits & Steel	1.33
5.	Blasting Supplies - Powder & Accessories	0.99
6.	Ground Support Supplies	0.42
7.	Pipe, Timber, Aux. Ventilation & Misc. Supplies	0.44
8.	Equipment Operating and Maintenance Supplies	0.18
	Subtotal	$17.30
	Misc. Costs @ 10%	1.73
	TOTAL	$19.03

Room & Pillar Stoping Cost Range $15.00 - $25.00

2.5 SELECTION OF MINE ACCESS AND HAULAGE METHOD

The selection of the mine access is a major decision in the planning of an underground ore body. This is the life line of the mine and will determine production capacity and flexibility, ease of service and access and have a major impact on operating and capital costs.

In choosing and designing the means of access the following design factors should be considered:

- Location of ore body in relation to topography
- Overburden depth and characteristics
- Depth of ore body
- Production tonnage requirements
- Capital costs
- Operating costs
- Ventilation requirements
- Probability of additional ore at greater depth

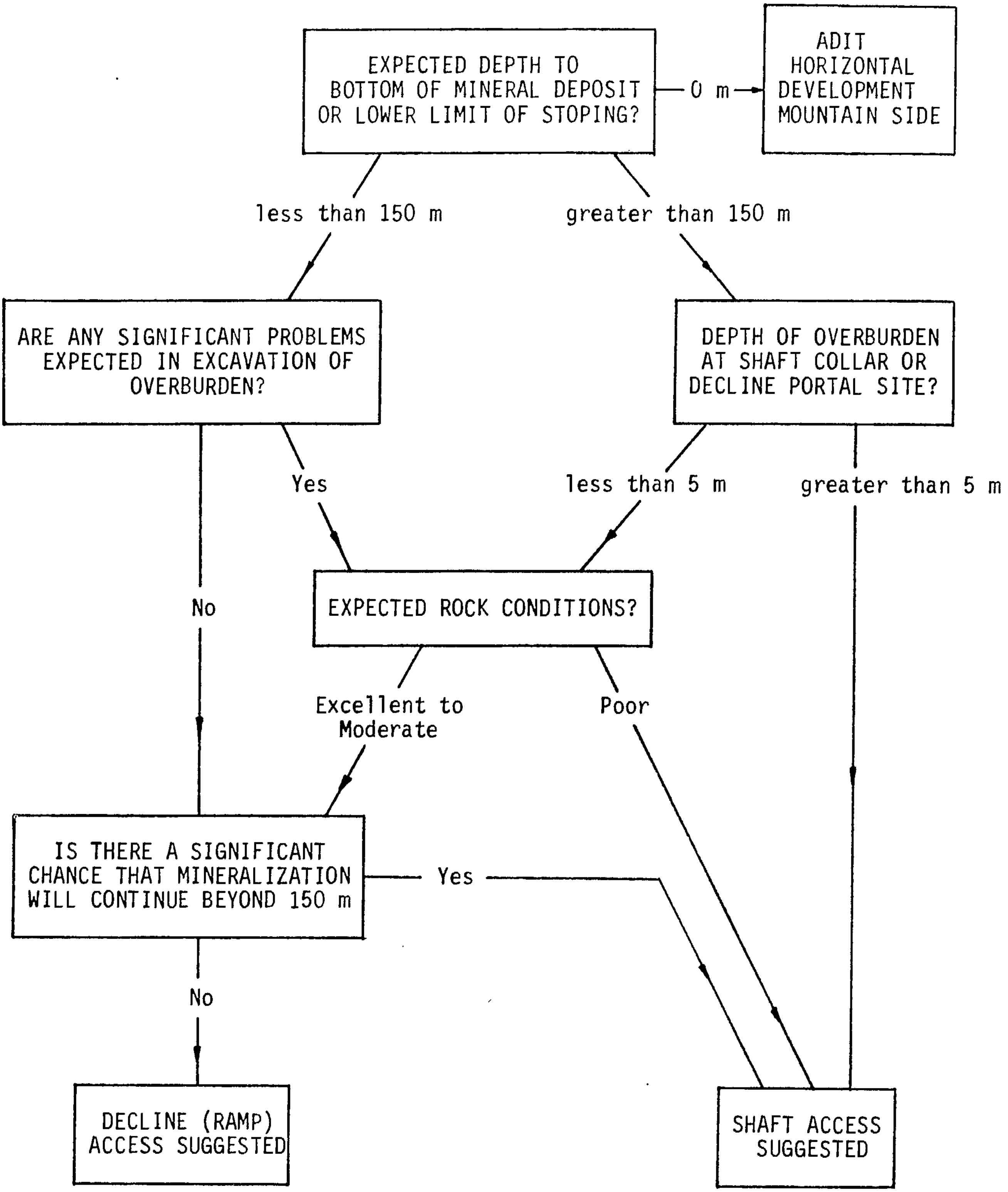
Three alternatives should be considered:

a) Shaft
b) Ramp
c) Adit

The access chosen will in part determine the underground haulage system chosen. Access/haulage combinations are listed below:

a) Shaft and track haulage
b) Shaft and trackless haulage
c) Ramp and trackless haulage
d) Adit and track haulage
e) Adit and trackless haulage

The following flow chart will allow the user to make a preliminary choice of means of access in order to determine anticipated operating and capital costs.


EXPECTED DEPTH TO BOTTOM OF MINERAL DEPOSIT OR LOWER LIMIT OF STOPING?
0 m
ADIT HORIZONTAL DEVELOPMENT MOUNTAIN SIDE
less than 150 m
greater than 150 m
ARE ANY SIGNIFICANT PROBLEMS EXPECTED IN EXCAVATION OF OVERBURDEN?
DEPTH OF OVERBURDEN AT SHAFT COLLAR OR DECLINE PORTAL SITE?
Yes
less than 5 m
greater than 5 m
No
EXPECTED ROCK CONDITIONS?
Excellent to Moderate
Poor
IS THERE A SIGNIFICANT CHANCE THAT MINERALIZATION WILL CONTINUE BEYOND 150 m
Yes
No
DECLINE (RAMP) ACCESS SUGGESTED
SHAFT ACCESS SUGGESTED

2.6 HOISTING AND RAMP HAULAGE

Costs per tonne identified in this section reflect the cost of moving the ore vertically from a loading pocket in a shaft, or from a fixed point along a ramp system. These costs include all direct operating labour, consumables and maintenance supplies.

Costs are determined by the choice of access alternative and the depth.

a) Shaft access with skip/cage hoist combination.

The operating costs for hoisting are largely fixed labour costs, i.e., the skip tender and hoistman are required at all times during mine operation. The hoisting cost/tonne can be optimized by maximizing the use of hoisting capacity on each working shift. The hoisting capacity/shift can be varied by selection of appropriate skip and hoist size and this selection should be made in consideration with the anticipated number of working shifts, expected production rate, depth of producing levels and some consideration of future expansion if required.

b) Ramp access with truck haulage.

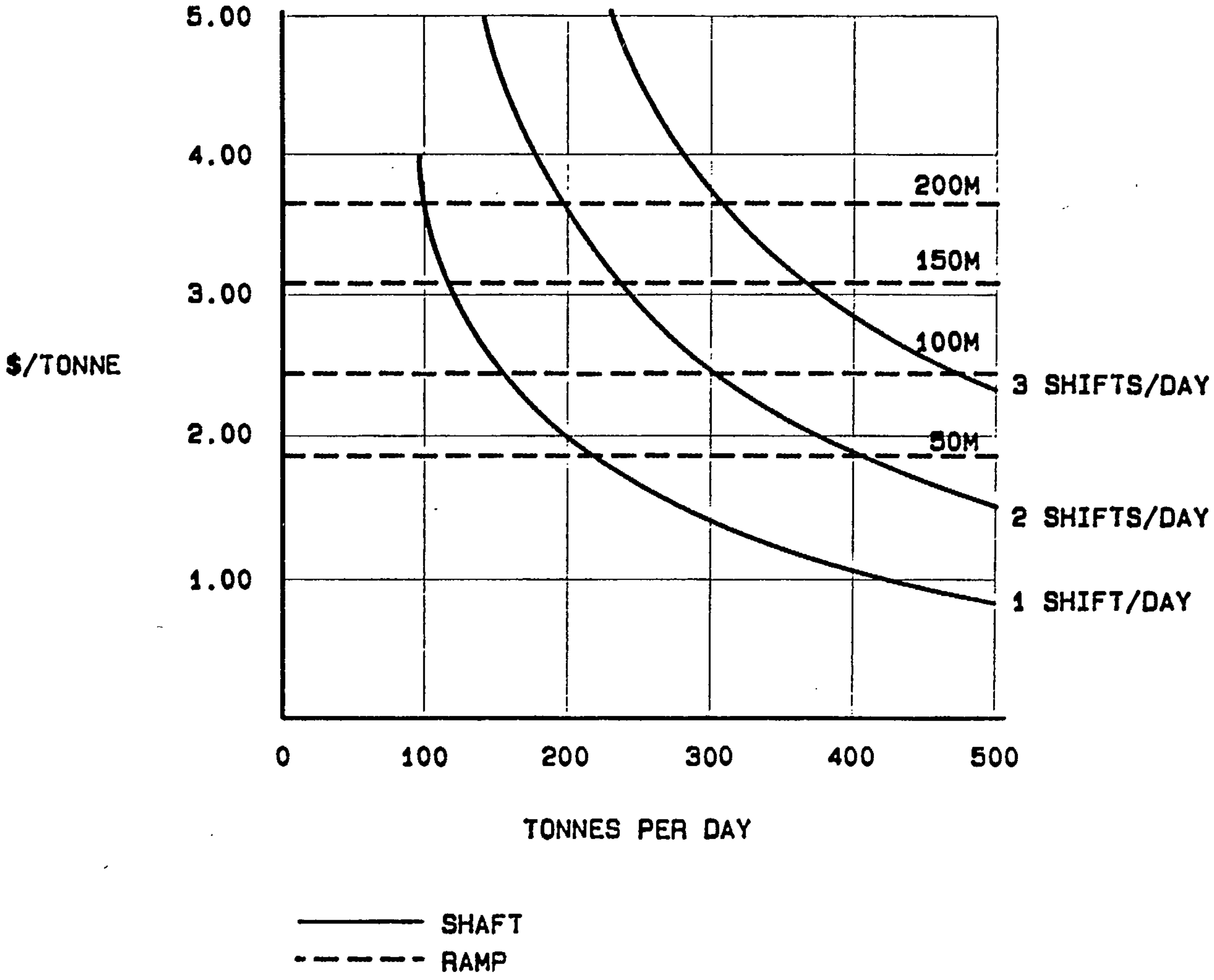
The operating cost/tonne for ramp haulage is governed by equipment operating cost/hour, labour cost/hour and haulage productivity which is related to the slope of ramp, the depth and the truck size.

For the purpose of this comparison an 11.8 tonne (13 ton) truck has been chosen to operate on a 15% ramp and the graph shows the cost/tonne at different haul depths. It has been assumed that the cost/tonne is affected only by the direct haulage costs i.e., operating, labour and maintenance costs are all directly related to tonnes hauled.

c) Adit haulage.

This is an extension of level haulage and the user should refer to Section 2.7 to determine these costs.

The following graph shows the hoisting costs/tonne for varying production rates for a one, two and three shift operation respectively; and the ramp haulage costs per tonne for various depths.



Manpower Schedule

For shaft hoisting, one hoistman and one skiptender should be allowed for each shift of operation.

For ramp haulage, the following manpower schedule is suggested for various production rates hauling from vertical depths up to 200 metres.

Vertical Haulage Depth	Production Rate (tonnes per day)				
	100	200	300	400	500
50 m	1	2	2	3	3
100 m	1	2	3	3	4
150 m	1	2	3	4	5
200 m	2	3	4	5	6

2.7 LEVEL HAULAGE

Costs per tonne identified in this section reflect the cost of moving the ore horizontally from a stoping area, or an ore pass, to the shaft or to the ramp. These costs include all direct operating labour, consumables and maintenance supplies.

The major decision to be made regarding level haulage concerns the choice of track or trackless methods.

Prior decisions made regarding mine access and/or stoping method may have already made, or at least limited, this choice. For example, if the mine is accessed by ramp, it is unlikely that track haulage will be used.

Assuming that the option is still open, some general comments pertaining to track and trackless haulage are listed below.

	Track	Trackless
Equip. Operating $/Hr. (excluding labour)	low	high
Payload	flexible	limited by drift size
Distance Hauled	equipment can be sized to suit haul distance	L.H.D. units very flexible and productive over short hauls. Trucks may be required for longer haul distances.
Ventilation Requirement	low	high
Loading and Dumping Time	high	low
Capital Costs	low	high

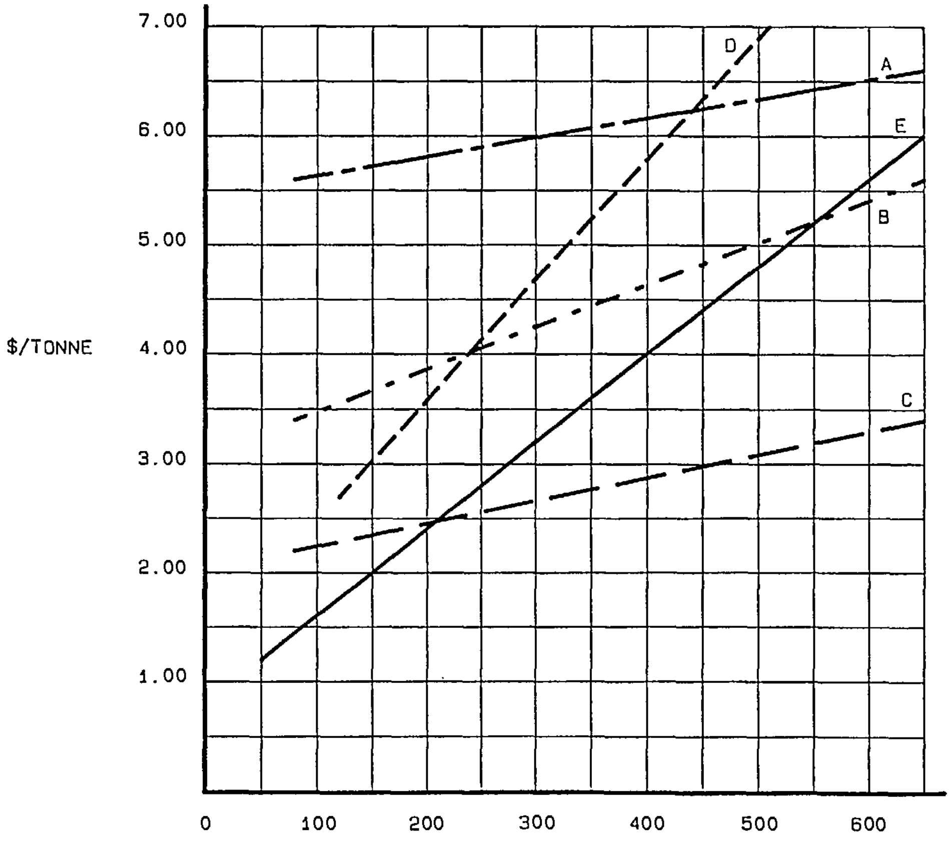
Costs

The three variables having the most significant impact on the operating cost of level haulage are:

i) the size and type of equipment;
ii) the method of loading;
iii) the distance hauled.

The graph below provides costs per tonne for five alternatives over a range of haulage distances.

LEVEL HAULAGE COSTS TRACK AND TRACKLESS



ONE WAY HAULAGE DISTANCE (METRES)

A. DRAWPOINT MUCKING WITH MUCKING MACHINE-12 TONNE TRAIN
B. CHUTE - 12 TONNE TRAIN (6X2 TONNE CARS)
C. CHUTE - 20 TONNE TRAIN (5X4 TONNE CARS)
D. DRAWPOINT MUCKING WITH 2YD^3 LHD (1.5m^3)
E. DRAWPOINT MUCKING WITH 3.5YD^3 LHD (2.7m^3)

Manpower Schedule

The following manpower schedule is suggested for various production rates for the five level haulage options.

Level Haulage Option	Production Rate (tonnes/day)				
	100	200	300	400	500
A	3	6	8	12	14
B	2	4	6	8	10
C	2	3	4	5	7
D	1	2	3	4	5
E	1	2	2	3	3

2.8 GENERAL MINE EXPENSE

"General Mine Expense" includes all labour and supplies not charged directly to a working place or activity such as development, stoping, level haulage or hoisting and generally includes:

- Maintenance labour for the underground facilities and equipment.
- Supplies handling.
- Misc. construction and level maintenance.

The major cost is labour and is proportional to the size of the active working area and installed services which are in turn proportional to the production rate.

The labour could be distributed to a working place and/or activity but because of the irregularity and/or general nature of the work, a general mine expense code is usually established. Labour would normally include:

- Underground maintenance crew (mechanics and electricians);
- Materials handling crew (nipping crew);
- General labour and construction crew.

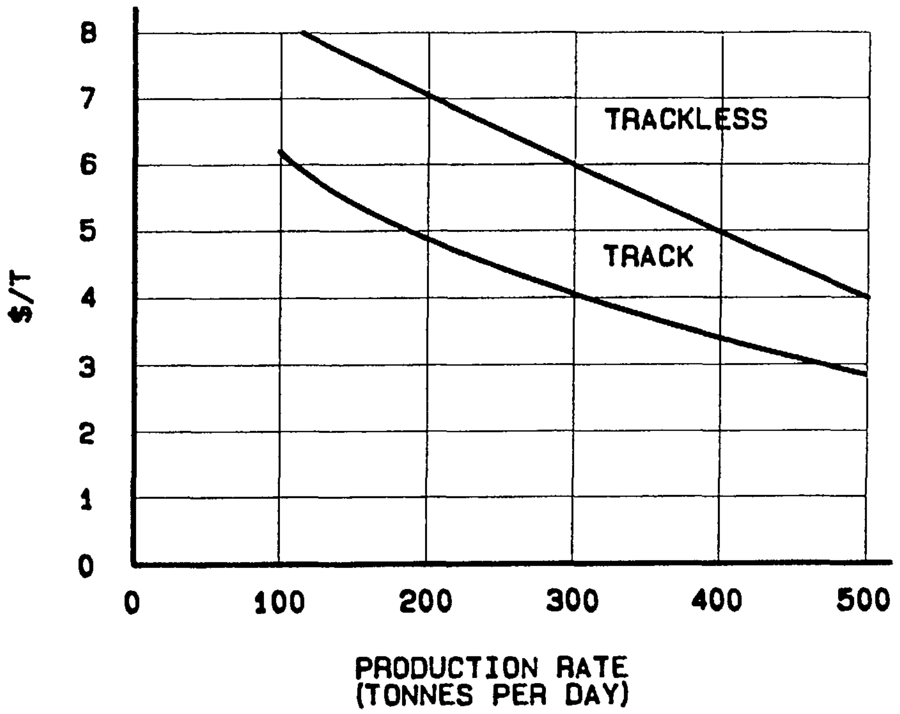
Each operation will be unique; however, the following manpower schedule is suggested for various production rates. These numbers should be adjusted to reflect any site specific circumstances.

Manpower Schedule

CLASSIFICATION	PRODUCTION RATE				
	100	200	300	400	500
U/G Mechanics/Electricians					
a) Track	1	2	3	3	3
b) Trackless	2	4	5	6	6
Nipping Crew					
a) Track	1	1	1	2	2
b) Trackless	1	1	2	2	2
General Labour & Const.					
a) Track	1	2	2	2	2
b) Trackless	1	2	2	2	2
TOTAL LABOUR PER DAY					
a) Track	3	5	6	7	7
b) Trackless	4	7	9	10	10

The following graph shows the cost/tonne of ore for both track and trackless mines at varying production rates. These costs are based on the above labour schedule, operating supplies for the nipping crew and minor construction material. The user should select the appropriate cost/tonne.

GENERAL MINE EXPENSE (COST/TONNE)



2.9 SURFACE PLANT AND MINE SERVICES

This section includes the cost of labour and materials to operate and maintain all surface plant and mine services including:

Surface Plant:

- Office and dry;
- Camp (bunkhouse/kitchen) - maintenance only
- Lamp room
- Surface shop and warehouse
- Yard and materials handling.

Mine Services:

- Electric power (diesel generation/hydro)
- Mine ventilation and heating
- Mine pumping and drainage
- Compressor house
- Water and sewage
- Access road maintenance.

Manpower Schedule

The following manpower schedule is suggested for various production rates:

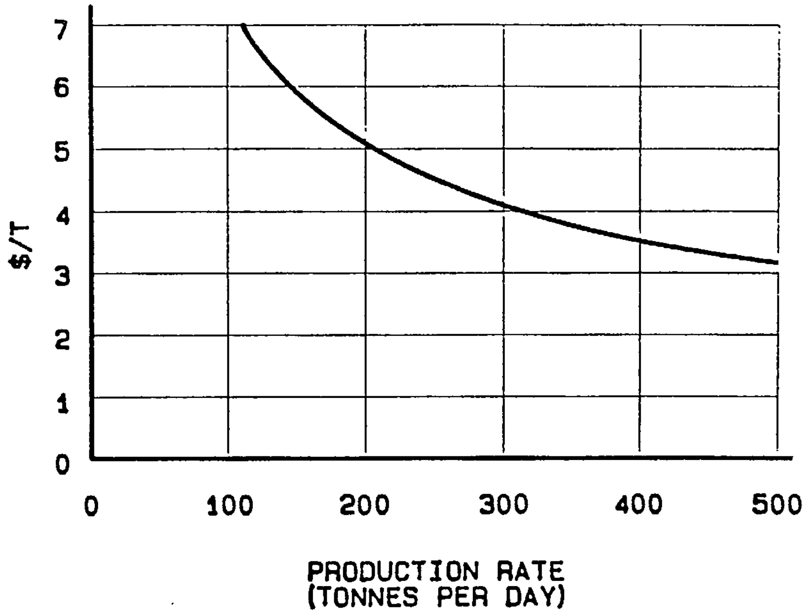
	PRODUCTION RATE				
CLASSIFICATION	100	200	300	400	500
Mechanics	2	3	4	4	4
Electricians	1	1	2	2	2
Equipment Operator	1	1	1	2	2
Labourer	-	-	-	1	1
Dryman	1	1	1	1	1
TOTAL*	5	6	8	10	10

* This does not include camp operating labour.

Costs are broken down into the following five items:

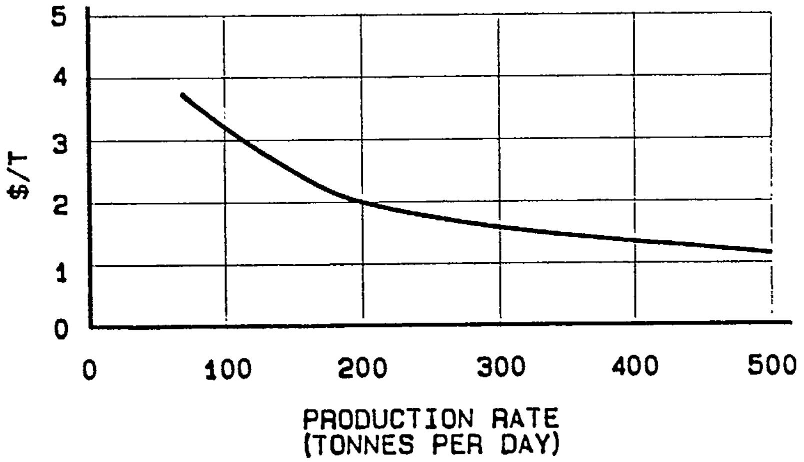
a) Labour

Labour costs indicated on the following graph are derived from the manpower schedule on the previous page.



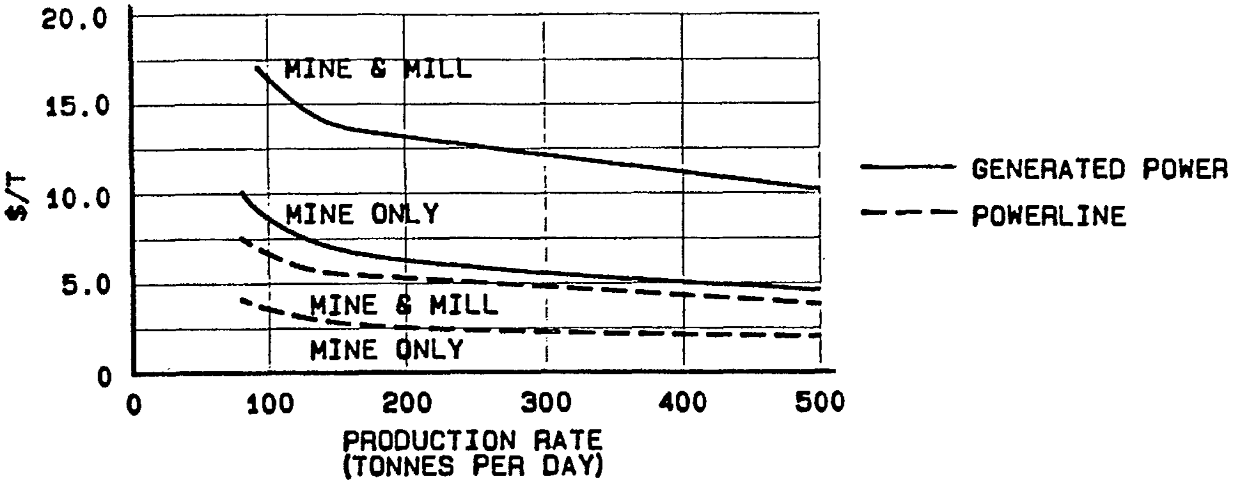
b) Materials and Operating Costs

The costs indicated on the following graph include operating and maintenance costs for compressors, fans, pumps and surface payloader as well as allowances for garbage and sanitary waste disposal and miscellaneous minor operating costs.



c) Power

Power costs are total property costs including, but not limited to, surface facilities, mine ventilation, mine hoist, compressors, pumps and mill.



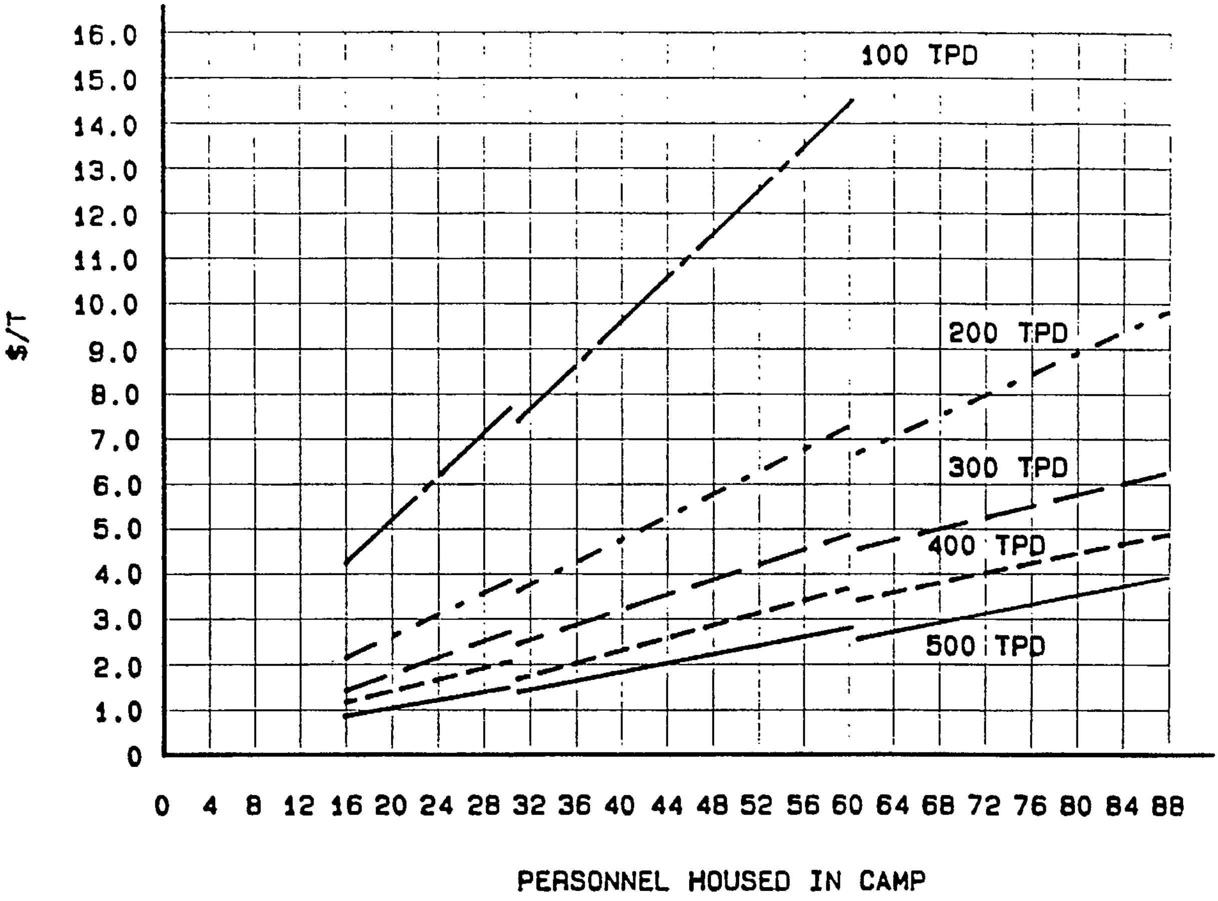
d) Camp

The camp costs indicated on the graph on the following page are based on the camp being operated by an outside caterer and include labour and operating supplies. Cost savings of approximately 30% can be realized if the mine operates the camp by employing their own cooks and cleaning staff.

Note:

A schedule of manpower is developed using Form 2(b). The percentage of employees housed in the camp will largely depend on the mine location. This percentage must be approximated by the user.

d) Camp Operating



e) Road Maintenance

A cost per tonne for road maintenance can be approximated by the following:

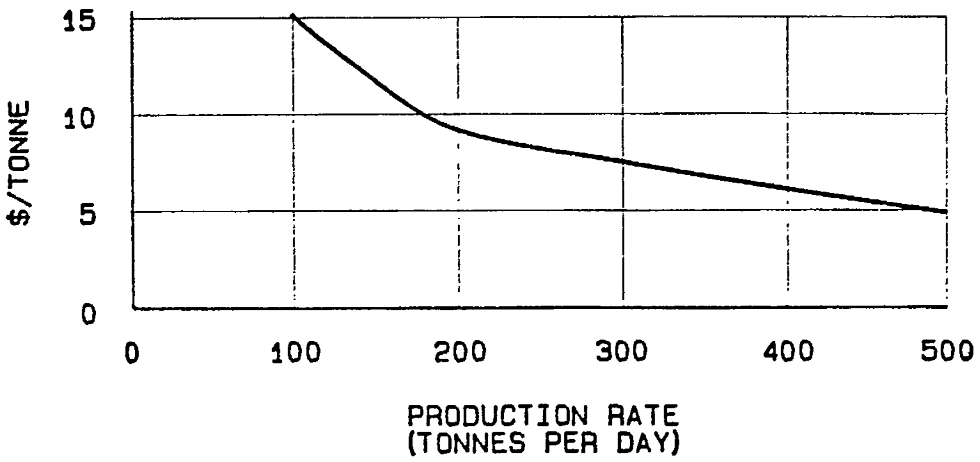
$$\text{Cost/tonne} = \frac{\text{Length of road (km) x \$500/km}}{\text{Production rate (t.p.d.) x working days/year}}$$

2.10 STAFF AND ADMINISTRATION

This includes all site supervision, support staff and operating supplies based on the following manpower schedule. The staff required will depend greatly on the number of working places.

	PRODUCTION RATE				
CLASSIFICATION	100	200	300	400	500
Manager	1	1	1	1	1
Mine Superintendent	-	-	1	1	1
Mine Engineer	1	1	1	1	1
Geologist	1	1	2	2	2
Surveyor/Technician	1	1	1	2	2
Accountant/Clerk	1	1	1	1	1
Shift Boss	1	2	2	2	2
Purchasing & Warehouse	1	1	1	1	1
TOTAL	7	8	10	11	11

The user can select an appropriate cost/tonne from the graph below and should adjust for any variances to the manpower schedule above.



2.11 MILLING

Milling costs can represent from 15% to 50% of total operating costs for small mines. Where there is a choice, the decision to process ore on-site or at a custom mill may have a significant impact on the viability of a mine. While the construction of a concentrator is a major capital expenditure, this may be more than offset by the impact of reduced milling costs and freight charges over the life of the mine. In some instances, it will no doubt prove to be cheaper to have the ore processed at an existing concentrator nearby.

To make the right decision, both alternatives should generally be costed and compared and consideration given to some of the advantages and disadvantages herein.

It is worth noting that the provision of an on-site concentrator will influence various elements of a minesite's infrastructure such as:

1) Environmental impact studies.
2) Site preparation area.
3) Tailings disposal.
4) Site services.
5) Power requirements.
6) Camp installation.

2.11.1 On-Site Milling

Some of the advantages and disadvantages of on-site milling are listed below:

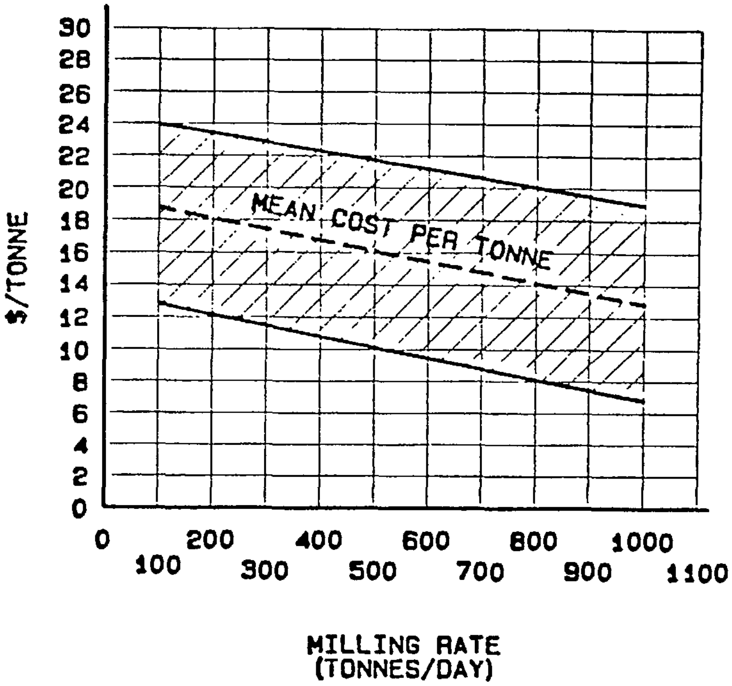
Advantages	Disadvantages
Milling cost per tonne should be lower.	Large up-front capital cost is required.
Owner has complete control of concentrating process.	Increased site services required.
Mill flow sheet can be tailored to mill feed for optimum recovery.	Tailings disposal is required.
Freight costs for transporting concentrate or bullion will be much lower than for crude ore.	Sales contracts for products must be arranged.
	Environmental requirements must be satisfied and permits obtained.

The following graph indicates a range of costs per tonne for mills with capacities ranging from 100 to 1,000 tonnes per day.

Costs will vary within the range depending largely on the amount of grinding required to liberate the mineral(s) and the complexity of the recovery process.

For example, ores that can be processed by straight cyanidation will cost less than those that require differential flotation in addition to cyanidation.

The costs indicated by the graph on the following page include staff and hourly labour, services, supplies and maintenance but <u>not</u> electrical power which is covered separately in Section 2.9.



Manpower Schedule

Description	Production Rate (tonnes/day) 100	200	300	400	500
Staff	3	3	4	5	5
Hourly	8	12	15	17	18
Total	11	15	19	22	23

2.11.2 Custom Milling

Some of the advantages and disadvantages of custom milling are listed below:

Advantages	Disadvantages
There is no capital expenditure required for the mill.	Freight costs can be high depending on distance.
There are no tailings disposal costs.	Recovery may be reduced as mill flow sheet not tailored to ore.
Site service requirements are reduced.	Owner may not get credit for 100% of the crude assay values.
	Custom milling may not be negotiable for the total mine life.

The cost of having ore custom milled will depend on a number of factors, the most significant of which are listed below:

a) The custom mill's basic cost per tonne. This will depend on the process, the mill's capacity, and the current milling rate.

b) The effect on the overall cost per tonne of milling additional tonnes.

   Ideally the additional tonnes will increase the mill's daily tonnage so that it matches the design capacity. In such a case the mill should operate at optimum efficiency and milling charges should be at the lower end of the scale.

   Conversely, if the mill requires modification to handle a different ore or expansion to handle the extra tonnes, custom milling charges are likely to be high.

c) The number of tonnes being custom milled.

d) Market conditions, that is the total custom milling capacity in a given area versus the demand for custom milling.

After considering these factors, select a custom milling cost per tonne from the range below.

Custom milling, cost per tonne	\$18.00 - \$30.00

Costs are all inclusive but do not include freight costs to transport ore to the mill.

2.12 MANPOWER SCHEDULE

Having developed a mine plan including, mining, ore handling, surface plant and services and staffing, it is a necessary and useful exercise to prepare a total manpower schedule for the operation. This will allow the user to determine an overall mine productivity which can be compared to similar operations as a check. Adjustments may have to be made at this stage. Use Form 2(b) to develop the manpower schedule.

Typical Example

The following manpower schedule is presented based on blasthole stoping, shaft access and hoisting, track haulage and a two shift mining and three shift milling operation for various production rates. The manpower requirements indicated for 'Ongoing Capital Development' are assumed for the purpose of the example only.

	Production Rate (Tonnes/Day)				
Description	100	200	300	400	500
Ongoing Capital Dev't. (avg)	1	1	1	2	2
Mining:					
Blasthole Stoping	2	5	7	9	11
Hoisting	4	4	4	4	4
Haulage	2	3	4	5	7
General Mine Expense	3	5	6	7	7
Total Underground	12	18	22	27	31
U/G Productivity (Tonnes/MS)	8.3	11.1	13.6	14.8	16.1
Surface:					
Plant and Services	5	6	8	10	10
Staff & Administration	7	8	10	11	11
Milling	11	15	19	22	23
Subtotal	23	29	37	43	44
Total On-Site Manpower	35	47	59	70	75
Overall Productivity (Tonnes/MS)	2.9	4.3	5.1	5.7	6.7

2.13 SUMMARY OF ON-SITE OPERATING COSTS

The final step in the development of operating costs is to pull all costs together in a summary of costs to provide a total on-site operating cost per tonne.

This cost would be used in the preliminary cash flow summary to assess the viability of the project.

Example

Total on-site operating costs are presented below for a mine/mill complex using blasthole stoping, shaft hoisting and track haulage. The mine operates on two shifts per day and the mill three shifts per day. A 20-tonne train is used to haul an average of 300 metres. Costs are developed for tonnages from 100 to 500 tonnes per day. Manpower is based on the schedule presented in Section 2.12. These costs do not include any depreciation or capital write-offs.

	Production Rate (Tonnes/Day)				
Description	100	200	300	400	500
Mining:					
Blasthole Stoping	11.11	11.11	11.11	11.11	11.11
Hoisting	7.30	3.60	2.50	1.90	1.50
Level Haulage	2.70	2.70	2.70	2.70	2.70
General Mine Expense	6.20	4.90	4.10	3.40	2.80
Total Underground	27.31	22.31	20.41	19.11	18.11
Surface:					
Plant and Services					
a) Labour	7.75	5.10	4.10	3.50	3.20
b) Material	3.25	2.00	1.60	1.30	1.10
c) Power (powerline)	6.75	5.25	4.75	4.50	4.00
d) Camp	8.60	5.90	4.70	4.10	3.50
e) Road Mtce.	Assumes no access road of significant length				
Subtotal	26.35	18.25	15.15	13.40	11.80
Staff & Administration:	15.20	9.00	7.50	6.00	4.90
Milling	18.75	18.00	17.50	17.00	16.00
TOTAL OPERATING COSTS/TONNE:	87.61	67.56	60.56	55.51	50.81

2.14 TRANSPORTATION OF MINE PRODUCT

On the following pages, haulage rates are provided for road and rail transport. It is assumed that haulage is not done by the mine operator. The rates do not include loading or unloading or transfer charges, intermediate stockpiling or warehousing, or insurances. The rates are typical of 1986 commercial rates in north-central Ontario.

Maps are included which show the location of non-ferrous smelters and refineries in Canada as well as the gold and silver producing areas. One of these locations will likely be the point of sale of the users final product be it untreated ore, concentrated ore, or precious metal bullion.

The user should recognize the tremendous impact that an on-site concentrator will have on his surface haulage costs. For example in a base metal operation where the mill head grades are 2% metal and the mill concentrate 25% metal, the tonnage of concentrate is approximately 1/13 of the mill feed tonnage.

In a precious metal operation, the cost of transporting bullion is insignificant and is not considered in this manual.

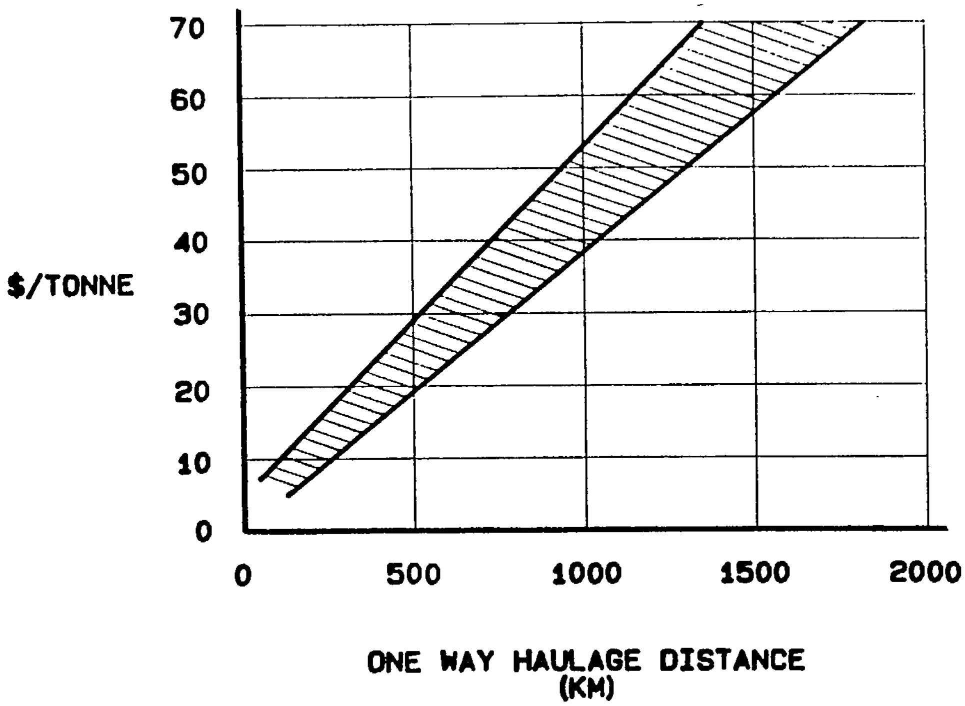
When using the road or rail haulage graphs on the next page to determine the appropriate cost per tonne mined for shipping concentrates, the cost selected from the graph must be divided by the concentrating ratio (C.R.). The latter is determined by dividing the grade of the concentrate by the mineable grade times the mill recovery or

$$\text{C.R.} = \frac{\text{Concentrate Grade}}{\text{Mineable Grade x Recovery}}$$

The mineable grade is determined on Form 5(e), and mill recovery factor on Form 5(f). Typical mill concentrate grades are listed below:

Metal	Typical Concentrate Grade
Copper	30%
Zinc	50%
Lead	60%
Other	5% to 90%

Rail Haulage (untreated ore and concentrates)

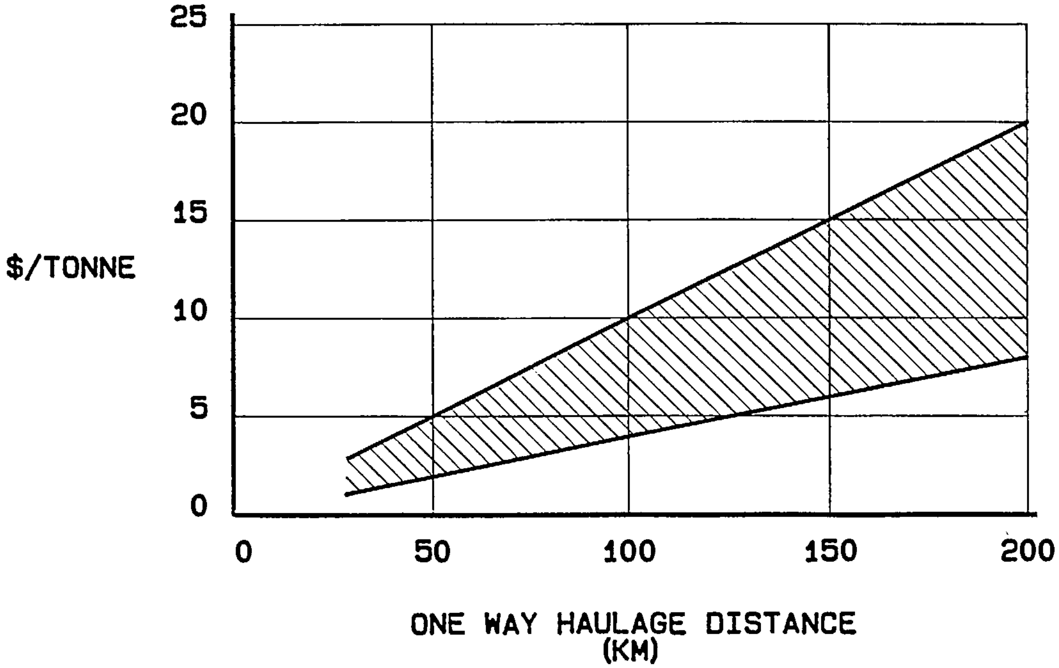


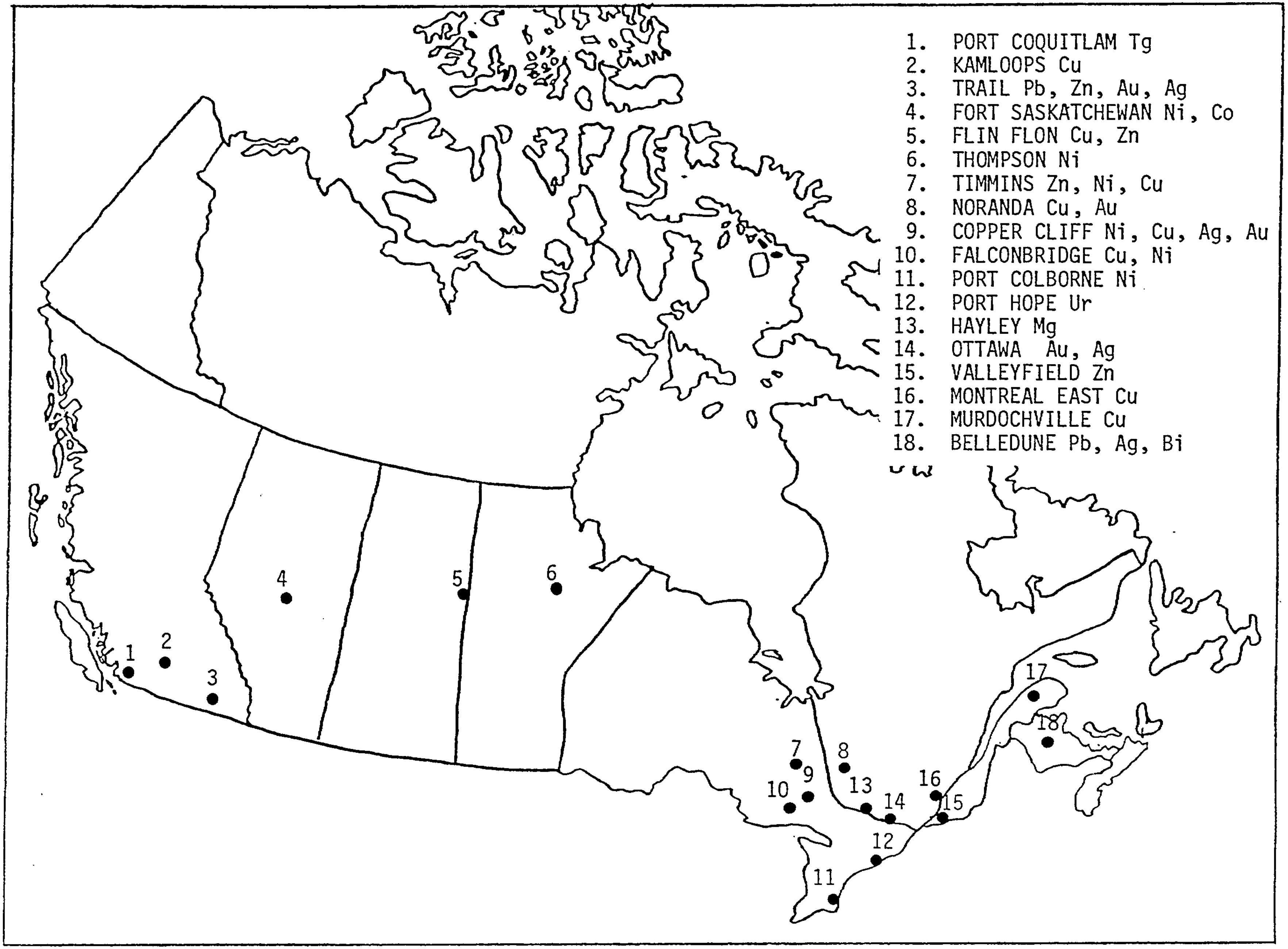
Road Haulage (untreated ore and concentrates)

Trucking costs will be influenced by two main factors.

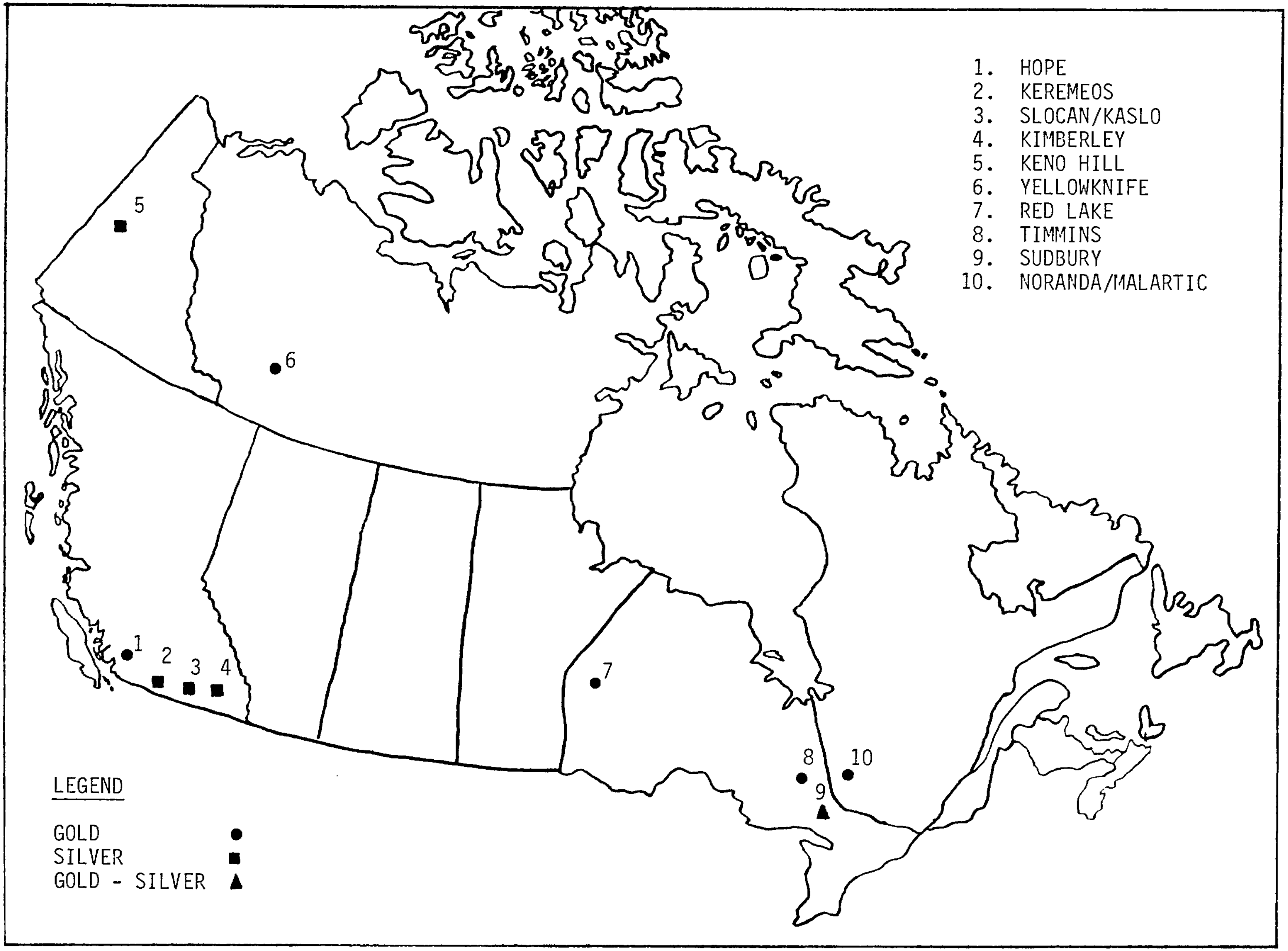
i) The ability to obtain return loads.
ii) The quality of roads over which the product is hauled.

Both factors are influenced by project location. Select a cost from the range presented accordingly.





CANADIAN NON-FERROUS SMELTERS AND REFINERIES



CANADIAN GOLD AND SILVER PRODUCING AREAS

SECTION 3

3.0 CAPITAL COSTS

3.1 INTRODUCTION AND CRITERIA

General

Capital costs, compiled using information contained in this section, should be entered and summarized on Forms 3 (a) and 3 (b). Form 3 (a) develops preproduction capital costs and Form 3 (b) operating capital costs.

Each subsection contains a description of the items covered therein. Should the user feel that the items covered differ from his requirements, a corresponding adjustment of costs can be made. When making such adjustments however, be careful not to eliminate an essential item or include an item covered elsewhere.

"Working Capital" or "Start-up Capital" has not been allowed for in this manual. The user should be aware, however, that there will be a period of time at the start of production when costs will be incurred with no revenue being received, and should budget accordingly. The time period will vary according to the method and location of the processing/refining facilities.

Cost Criteria

The following criteria have been used in developing Capital Costs:

1) Costs are in first quarter 1986 Canadian dollars.

2) Preproduction development and construction is to be done by a contractor, on a three shift, seven days per week basis.

3) Capital costs are estimated for a property located in north - central Ontario. Capital costs are to be adjusted to suit the actual property location through the use of Regional Cost Factors included in Section 4.0.

4) Capital costs are all-inclusive and cover procurement, transportation and installation costs. Preproduction development costs include the cost of labour, equipment write-offs, and consumable supplies including electric power and compressed air.

5) All units are metric.

6) In some instances a project may have a special requirement that is not addressed by this manual. Some special requirements not addressed by this manual include:

   a) shaft freezing
   b) tunnel boring
   c) radiation protection.

7) Costs presented assume that road access is either existing, or will be constructed prior to on-site construction and excavation.

3.2 FEASIBILITY STUDIES AND DETAILED ENGINEERING

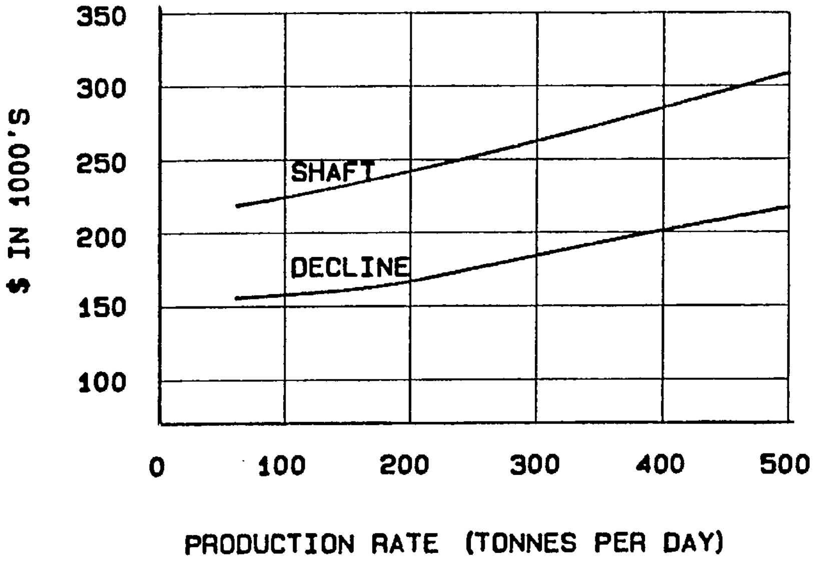
General

This section covers the cost of:

i) prefeasibility studies;
ii) feasibility studies;
iii) detailed engineering.

Costs

By using the production rate and means of mine access previously determined, select the appropriate cost from the graph below:



The above graph includes costs for the following:

i) Mineral reserve estimates;
ii) Evaluation of alternatives;
iii) Mine planning and scheduling;
iv) Metallurgical evaluation;
v) Tender document preparation.

3.3 ADDITIONAL DIAMOND DRILLING AND SAMPLING

General

This section covers the cost of additional drilling and sampling that will be carried out prior to the start of production.

Costs are included for:

i) Diamond drilling from surface;
ii) Underground diamond drilling;
iii) Assaying of samples.

Should a drilling program already be outlined, or if the user can estimate the quantity of drilling required, use those figures. If not, refer to the guidelines below to determine the number of holes to be drilled. The user is required to estimate the length of holes from the existing knowledge of the geometry and depth of the deposit.

3.3.1 Drilling from Surface

a) Number of holes

Allow for one hole every 30 metres of strike length.

b) Costs (B-size core)

Basic cost	$65.00/metre
Additional costs:	
Drilling from ice	$10.00/metre
Hard, abrasive rock	$10.00/metre

3.3.2 Underground Drilling

a) Number of holes

Allow three holes (one up, one flat, one down) every 30 metres of strike length for drilling on initial production level.

Subsequent definition diamond drilling is covered under operating costs.

b) Costs (B-size core)

Basic cost	$45.00/metre
Additional costs:	
Hard abrasive rock	$10.00/metre

3.3.3 Assaying Samples

Assaying sample for one element	$12.00
Assaying for additional elements	$ 5.00 each

Direct assaying costs may be reduced if assaying is carried out on-site. This option has not been considered in this manual.

3.4 PERMITS AND ENVIRONMENTAL STUDIES

General

This section covers the cost of:

a) obtaining necessary permits;

b) undertaking environmental studies, should they be required.

Costs

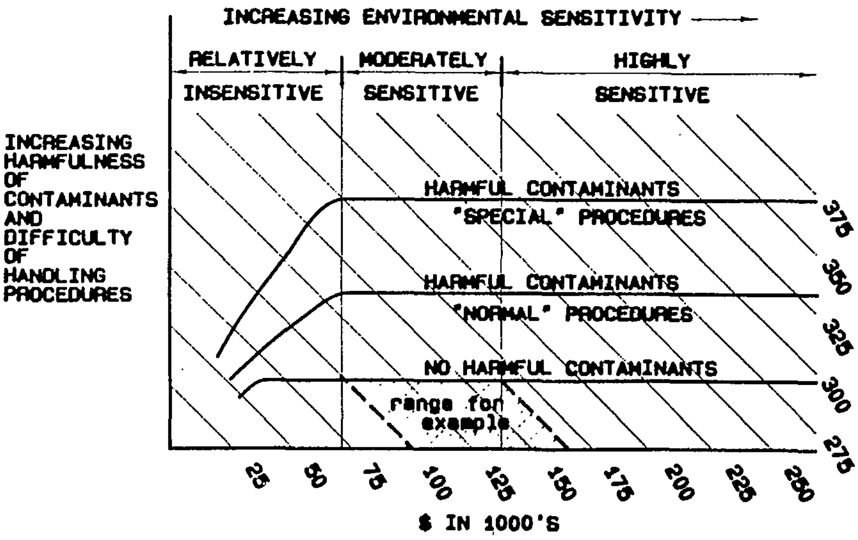
The time and cost required to obtain permits and carry out environmental impact studies are both related to:

a) the environmental sensitivity of the region;

b) the amount and type of contaminants that will be produced and the complexity of the procedures required to control them.

The graph below approximates the effect that these two elements have on costs. Be aware, however, that individual circumstances can cause costs to vary significantly from those presented.

As an example, a mine located in a moderately sensitive area, producing no harmful contaminants, can expect costs ranging from $100,000 to $165,000.



3.5 PROJECT MANAGEMENT AND PREPRODUCTION SCHEDULING

General

This section covers the Owner's on-site costs during the preproduction phase of the project.

These include:

i) Owner's site representative;
ii) Owner's site geologist;
iii) Operating and administration costs incurred by on-site personnel during the preproduction phase.

Costs

The total cost of project management prior to production start-up is a function of:

i) the average monthly cost;
ii) the duration of the preproduction phase.

Guidelines are offered below for calculating each of these items. The user is required to calculate each item and multiply together to arrive at a total cost.

Example of Average Monthly Cost

	$/Month
Site Engineer (salary + burdens + living costs)	5,500
Site Geologist (salary + burdens + living costs) (50% of time)	2,500
Pickup/telephone/office & misc. costs	1,500
Total	9,500

Parameters for Estimating Preproduction Schedule

Suggested schedule periods and rates of advance are listed below for both ramp and shaft mines. The user is required to determine the duration of preproduction development and construction based on these guidelines.

		Ramp	Shaft
i)	Mobilization and Surface Setup		
	Select contractor, mobilize, setup & complete surface work prior to excavation of ramp or shaft.	2 months	4 months
ii)	Excavation of Mine Access		
	Vertical advance rate	30 metres/month	75 metres/month
	Schedule:		
	Depth ÷ vertical advance rate (eg. vertical depth of 150 m)	(5 months)	(2 months)
iii)	Excavate Shaft Stations	N/A	¼ month each
iv)	Ancillary Shaft Excavations & Installations	N/A	1 month
v)	Preproduction Development		
	Advance rate - track	N/A	180 metres/month
	(metres/level) - trackless	240 metres/month	240 metres/month
	Schedule: (refer to Section 3.16)		
	Development for 2 years mining ÷ (advance rate/level x # of levels)		
vi)	Miscellaneous Construction	concurrent with other work	
vii)	Diamond Drilling		
	Includes evaluation of drill core & mine planning	3 months	3 months

Note: The above schedule assumes continuous work.

3.6 ACCESS TO MINESITE

General

This section covers the cost of establishing a means of access to the site.

It includes:

i) New road construction;
ii) Upgrading existing roads;
iii) Road bridges.

As stated in the capital cost criteria, costs included in this manual assume that road access is either existing, or will be constructed prior to on-site construction and excavation. The following alternatives are included for reference only.

i) New railway spur lines;
ii) Barges and docks;
iii) Remote airstrips;
iv) Winter roads.

3.6.1 New Road Construction

a) Road Length

The cost range presented in Section b) assumes that the road is routed to avoid 'problem areas' that would cause costs to become excessive.

These 'problem areas' may include:

i) routes requiring more than occasional rock excavation;
ii) regions with numerous water courses;
iii) swamps;
iv) excessive changes in grade.

The user should keep points i) to iv) above in mind when estimating the length of the access road.

b) Costs

The costs below allow for clearing, grubbing, fill, granular material and occasional culverts to construct a gravel road of sufficient width to allow trucks to pass with one vehicle stationary.

'Typical' access roads will cost \$75,000 - \$125,000 per kilometre with \$100,000 being a reasonable average.

If sections of the road cannot avoid difficult terrain an additional cost allowance should be added accordingly.

3.6.2 Upgrading Existing Roads

Costs will naturally depend on the state of the existing road.

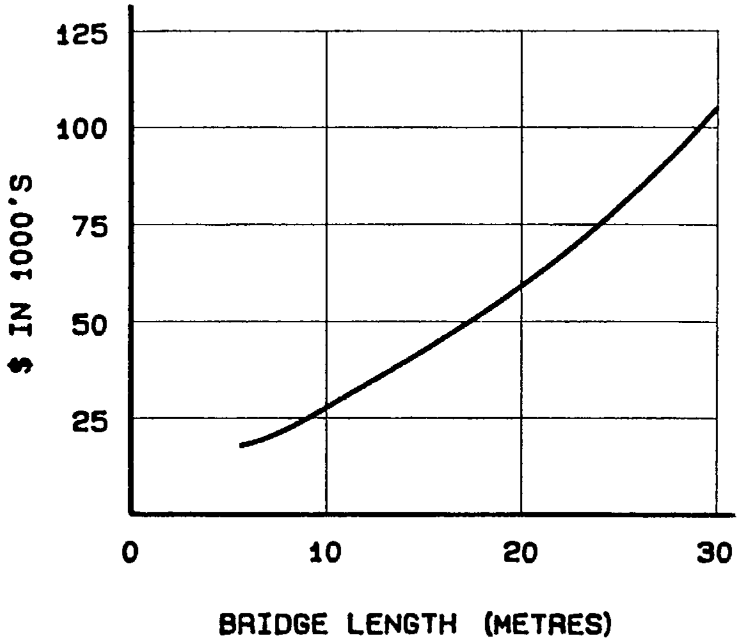
Upgrading may only consist of improving the road surface or may involve widening, replacing culverts and straightening curves in addition to re-surfacing.

Estimate the cost of work required within the range of 15% - 50% of the cost of new road construction.

ie) \$15,000 - \$50,000 per km.

3.6.3 Road Bridges

The graph below presents costs for single lane bridges up to 30 metres in length. It assumes that there are no significant problems with foundations.



3.6.4 New Railway Spur Lines

Unit costs are tabulated below for new railway spur lines. Train bridge costs are not included.

Type of Terrain	Description	Cost
1	Flat terrain. Fill is required over rock outcrops along spur line route.	$510,000/km
2	Moderately flat terrain. Some fill and rock blasting is required along spur line route.	$550,000/km
3	Flat terrain. A significant amount of fill is required to traverse muskeg along spur line route.	$950,000/km

3.6.5 Barges and Docks

Should the location of a mine site require that a road be constructed around a lake, a barging system may offer a more economic alternative. The range of costs indicated below allows for a motorized barge, timber and rock fill docking facilities at two locations, a motor boat for personnel transport and lifting devices. Costs will largely depend on the suitability of dock sites and the size of barge required.

Estimated cost range $150,000 - $300,000

3.6.6 Remote Airstrip

The costs of establishing a remote private use airstrip are summarized in the table below:

GRAVEL AIRSTRIP COSTS

Type of Aircraft to be Used	Approximate Runway Length	Cost to Construct Remote Gravel Airstrip
DC-3, Twin Otter or equivalent	1100 m	$410,000
737, Hercules or equivalent	1600 m	$515,000

Note: The costs presented above assume terrain "reasonably suited" to the construction of an airstrip and a nearby source of good gravel.

3.6.7 Winter Roads

Allow $1500 per kilometre for the initial construction of a winter road. This cost includes initial helicopter reconnaisance, site clearing, and winter road construction.

3.7 SITE PREPARATION

General

This section deals with the cost of providing an adequately level and free draining site for construction. The work undertaken may include:

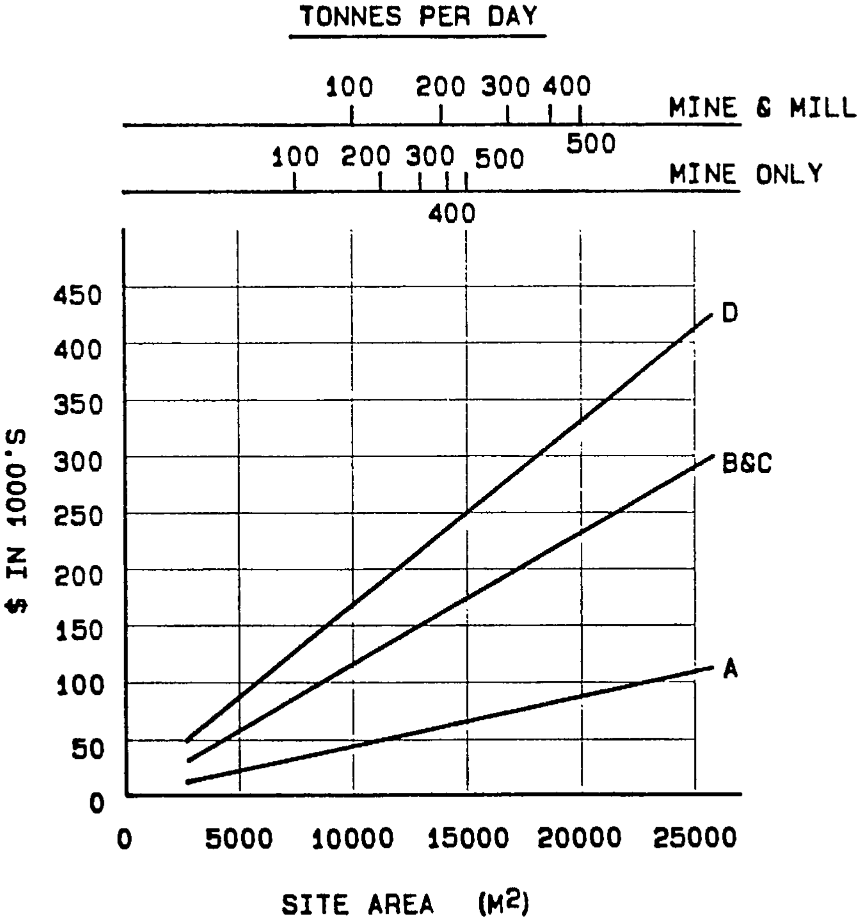
i) Clearing and grubbing;
ii) Overburden stripping;
iii) Filling and grading;
iv) Blasting, levelling, and terracing;
v) Placement of geotextile materials;
vi) Site drainage.

Costs

Costs will depend on the size of the area to be prepared, and the topography and surface ground conditions found at the site.

Identify the site category (or mixture of categories) that best describes the site being evaluated.

The graph on the following page relates costs for a range of areas in each site category. Site areas of mines and mine and mill complexes are also approximated for varying production rates.



Site Category	Description
A	Ideal site - flat, dry, and free draining.
B	Uneven ground with rock outcrops.
C	Flat ground with muskeg.
D	25% rock slope requiring terracing.

For sites having mixed topographic conditions:

1) Determine the site area required for the production rate.
2) Estimate area of each topographic category.
3) Combine costs of each topographic area.

3.8 CAMP INSTALLATION

General

This section covers the cost of supplying and installing a camp facility for the Production Phase Only. The costs presented for pre-production work include the installation, rental and operation of a temporary camp.

The camp facility includes:

i) Sleep trailers
ii) Wash and toilet facilities
iii) Kitchen and dining facilities
iv) Limited recreational facilities

Camp Capacity

The capacity of the camp required will depend on:

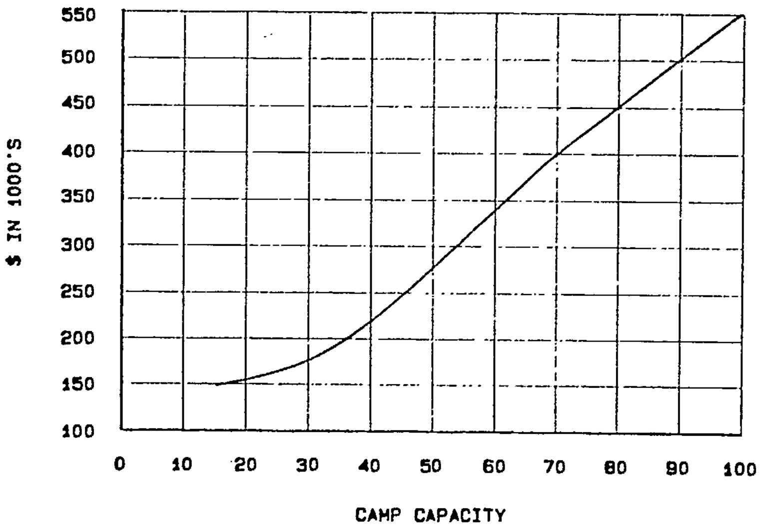
i) Total on-site manpower - refer to Form 2(b).

ii) The percentage of the on-site manpower that requires housing in a camp. This will depend on the mine location and must be approximated by the user.

After considering the two points above, an allowance of an additional 10% should be included to account for fluctuations in manpower.

Costs

Refer to the graph below and select a cost based on camp capacity. Costs assume the use of trailer modules which will allow flexibility, ease of installation and expansion.



3.9 SITE SERVICES

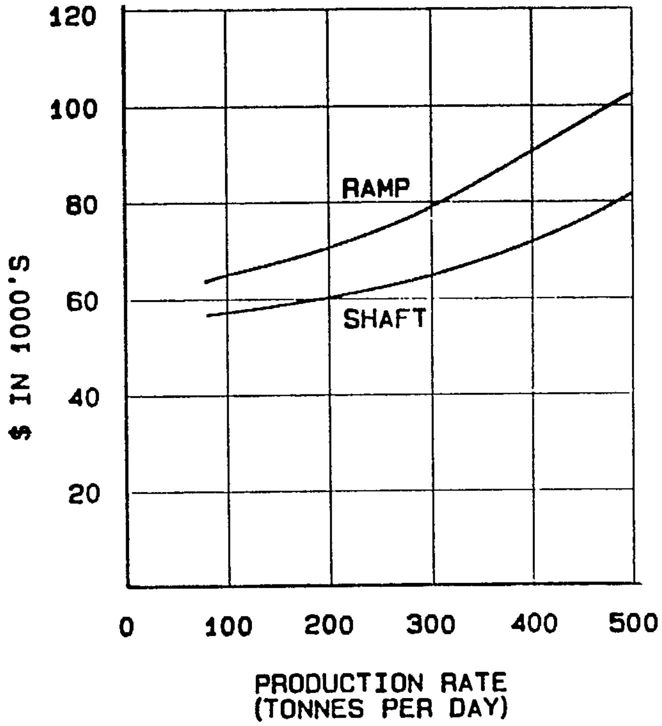
General

This section covers the provision of all site services with the exception of electrical power and compressed air. Site services include the following:

i) Potable and process water supply.
ii) Sewage handling.
iii) Communications - telephone and telex.
iv) Fuel storage.
v) Storage areas.

Costs

The graph below illustrates the relationship between the capital cost for site services and the mining rate for both shaft and ramp/adit access. Select a capital cost based on the mine production rate, and means of mine access to be used.



3.10 ELECTRICAL POWER AND COMPRESSED AIR

General

This section covers the supply and surface distribution of electrical power and compressed air.

Electrical power may be supplied from either a powerline or generators depending on the availability of a suitable powerline.

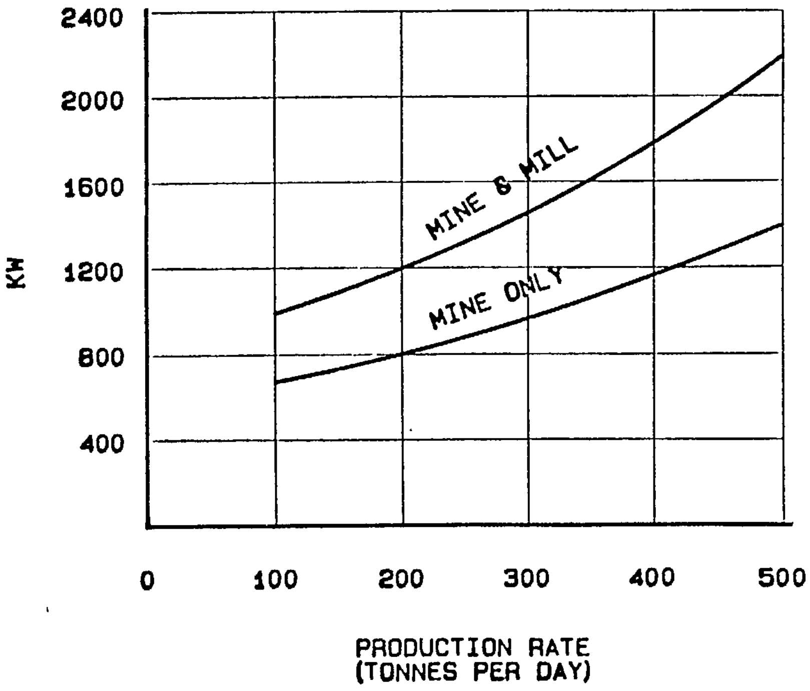
Compressed air is considered to be supplied from electric compressors.

3.10.1 Electrical Power

a) Electrical Power Capacity

The graph below approximates the power capacity required by "typical" mines and mine/mill complexes.

The user should be aware that the amount of power actually required will vary depending on hoist size, ventilation requirements, pumping capacity, type of mill, etc.



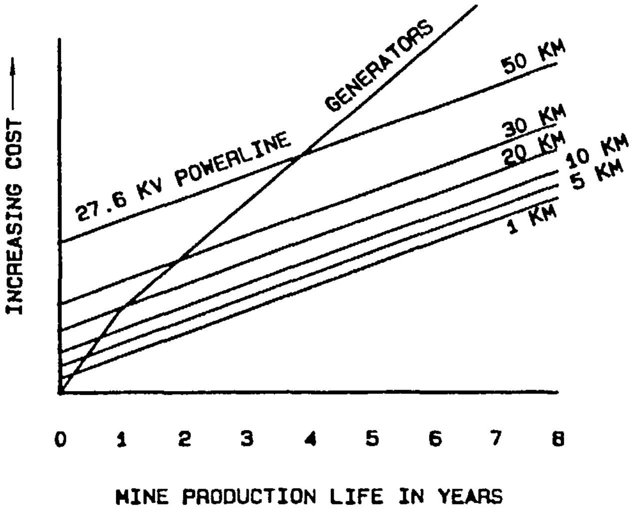
b) Powerline or Generators?

i) Is a powerline with free capacity available? If so, at what distance from the mine site? If the answer is unknown, contact the local power authority.

ii) If power is available, is it more economical to construct a powerline or operate with generators?

The following graph approximates relative power costs for generators and powerlines of various lengths. It assumes a total connected load of 1000 kW.

Starting with the expected mine life, determine which is the most economical choice.



Note:

i) Loads less than 1000 kW will improve the relative position of generators and vice-versa.

ii) The fact that up-front capital costs can be reduced by using generators may outweigh a slightly higher overall cost.

iii) By guaranteeing a minimum power demand over a period of years, the cost of a powerline can sometimes be recouped in reduced usage charges.

c) Costs

i) Powerline

Costs are split into two areas:

- Powerline - pole line and conductor (dependent on length and voltage)
- Site costs - transformer station and surface distribution

  (dependent on voltage & site specifics)

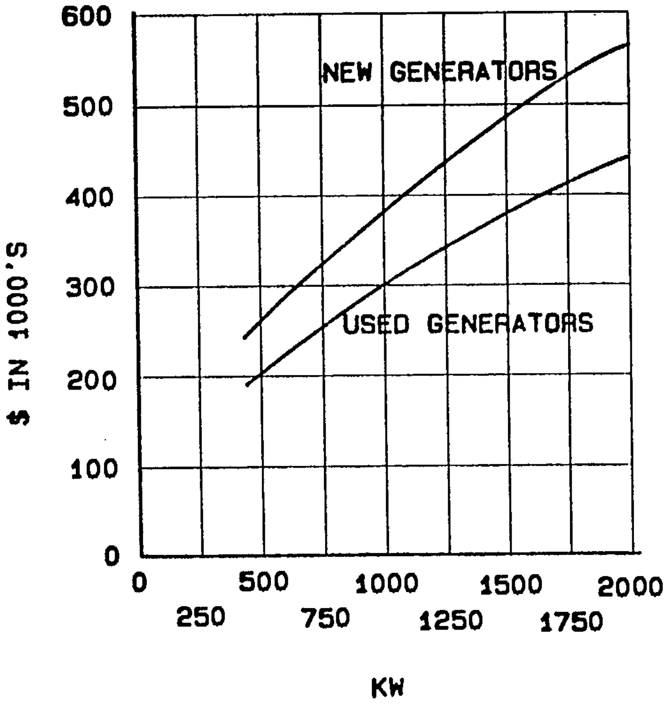
Assuming 27.6 kV:

Powerline cost	- $30,000 per Km
Site cost	- $200,000

Costs will increase somewhat at higher voltages.

ii) Generating Plant

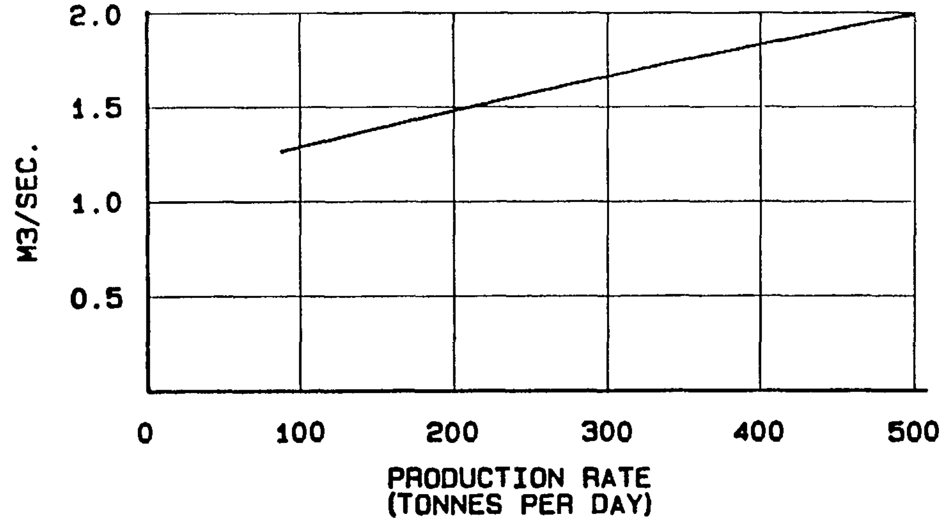
The costs indicated on the graph below include the supply and installation of generators and surface distribution of electrical power. Costs allow for a spare generator.



3.10.2 Compressed Air Plant

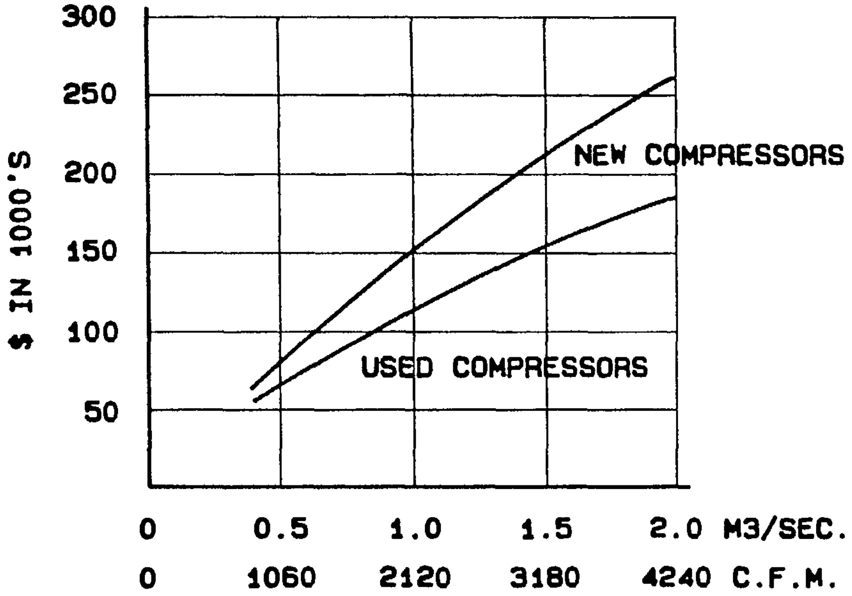
a) Compressed Air Capacity

Determine the capacity required from the graph below:



b) Costs

The graph below indicates the cost of the supply and installation of a compressor plant. Identify costs based on the plant capacity determined above.



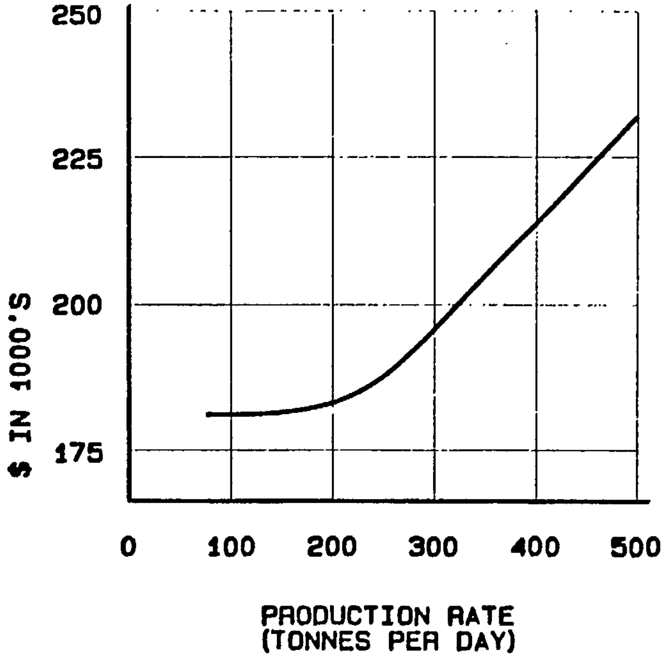
3.11 OFFICES, SHOPS, DRY, WAREHOUSE

General

This section covers the capital cost for the supply and erection of the permanent office, shop, dry and warehouse facilities required for the production phase of the project. Cost estimates for pre-production construction and excavation allow for the installation, rental and operation of temporary facilities.

Costs

Refer to the graph below and select the capital cost based on the expected mine production rate.



The costs presented are based on the following:

i) Buildings are new prefabricated, or pre-engineered, and trailers are utilized where suitable.

ii) All buildings are equipped for operation:

- a) Office complete with desks and filing cabinets.
- b) Shop complete with overhead crane, work benches, etc.
- c) Dry complete with lockers, hangers and wash facilities.
- d) Warehouse complete with shelving.

3.12 MINE ACCESS

General

This section covers the cost of providing a means of access to the deposit at the required elevations.

This can be by shaft, decline or adit(s).

All-inclusive contractor costs are provided for:

i) Shafts
  - mobilize, setup, teardown, demobilize
  - shaft collar (in rock and overburden)
  - shaft excavation and equipping
  - shaft changeover to skipping

  Note: Owner's hoist and headframe are utilized for shaft sinking.

ii) Declines
  - mobilize, setup, teardown, demobilize
  - decline portals (in rock and overburden)
  - decline excavation

iii) Adits
  - mobilize, setup, teardown, demobilize (as per decline)
  - portals (as per decline)
  - adit excavation and internal ramps

3.12.1 Shafts

a) Mobilize, Setup, Teardown, Demobilize

To mobilize contractor to site, set up temporary surface facilities, install sinking gear prior to start of shaft sinking and teardown and demobilize once shaft sinking is completed.

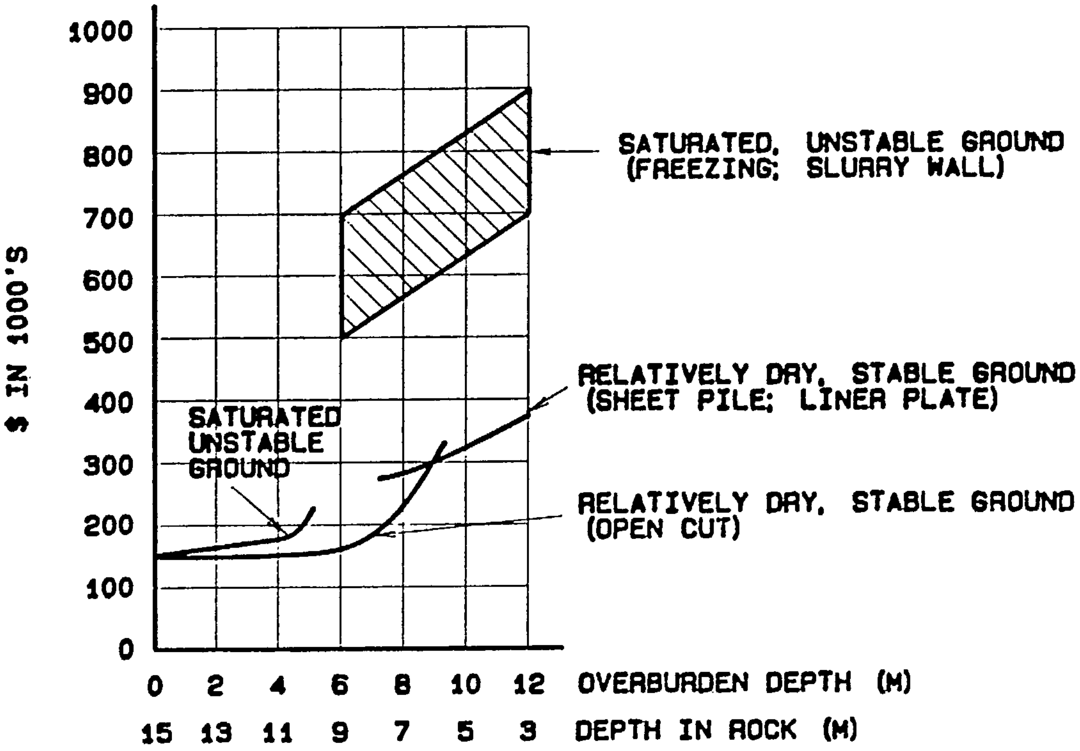
Allow one lump sum	$225,000

b) Shaft Collars

Shaft collars are assumed to 15 metres in depth in all cases.

The graph below presents costs for collars using various excavation techniques. The choice of technique, and therefore costs, will depend on the depth, type and water content of the overburden.

Identify the cost corresponding to the overburden depth (if any) and ground conditions that best describe the project site.



c) Shaft Excavation and Equipping

Costs are presented for:

- ° 2-compartment timber shafts 1.83m x 1.83m compts. (6'x6')
- ° 3-compartment timber shafts 1.83m x 1.83m compts. (6'x6')
- ° 4.5 m diameter concrete shafts

It is likely that a timber shaft will be selected over concrete unless poor or squeezing ground is anticipated.

The choice of 2 or 3-compartments will depend largely on hoisting requirements. Refer to Section 3.14, "Hoisting Systems" and also consider the possibility of future increases in production rate.

When calculating shaft depth:

i) Allow 30 metres past the bottom level for loading pocket.

ii) Subtract collar depth (15 metres) from total depth to determine shaft length.

The graphs presented on the following pages indicate costs for each of the shaft alternatives.

Costs allow for excavation, equipping, ground support and installation of bearing sets, catch pits and water rings. Shafts requiring freezing or extensive grouting are outside the scope of the graphs presented.

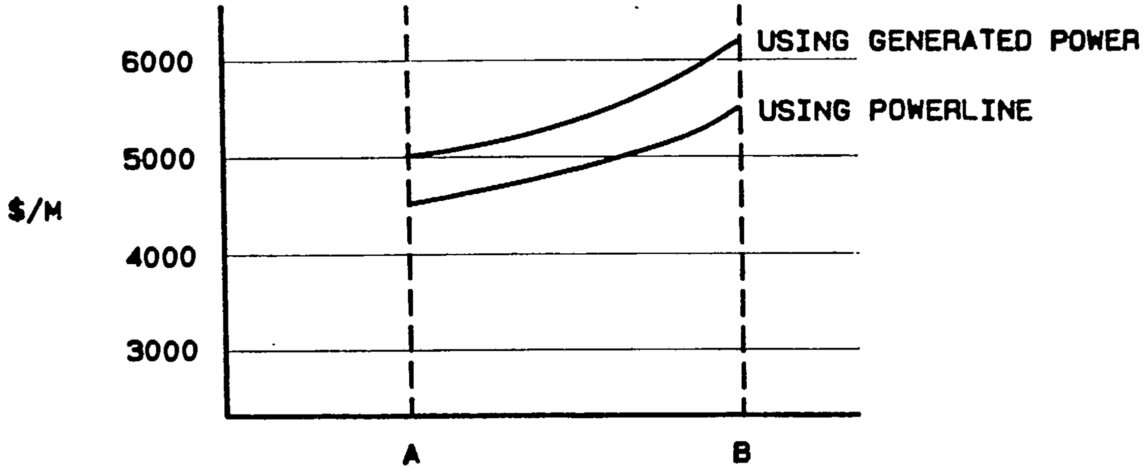
c) Shaft Excavation and Equipping (Continued)

Costs are presented for a range of ground conditions ranging from A to B.

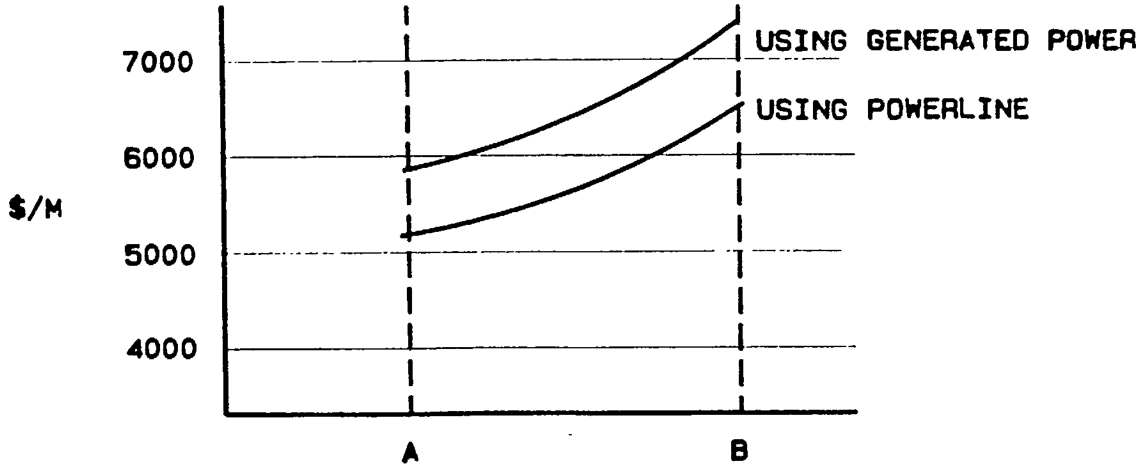
A = Good ground, random bolting only.

B = Poor ground, bolting to bench affecting excavation performance.

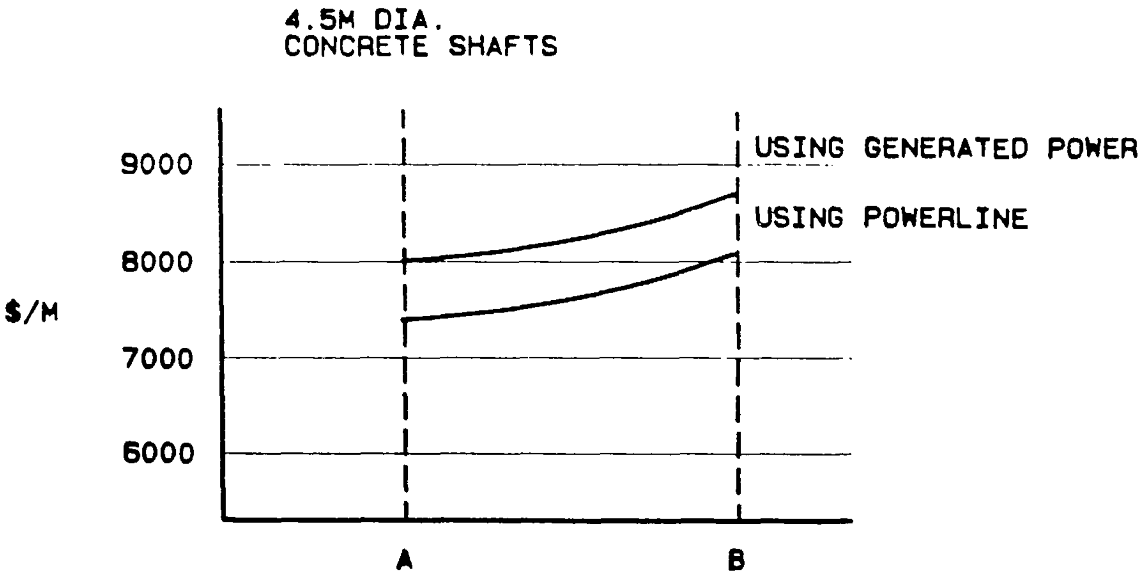
TWO COMPARTMENT TIMBER SHAFTS



THREE COMPARTMENT TIMBER SHAFTS



c) Shaft Excavation and Equipping (Continued)



d) Shaft Changeover to Skipping

Costs include installing permanent ropes, suspending skip(s) and cage, testing and commissioning system.

Allow one lump sum - $40,000

3.12.2 Declines

a) Mobilize, Setup, Teardown, Demobilize

To mobilize contractor to site, set up temporary surface facilities, teardown and demobilize.

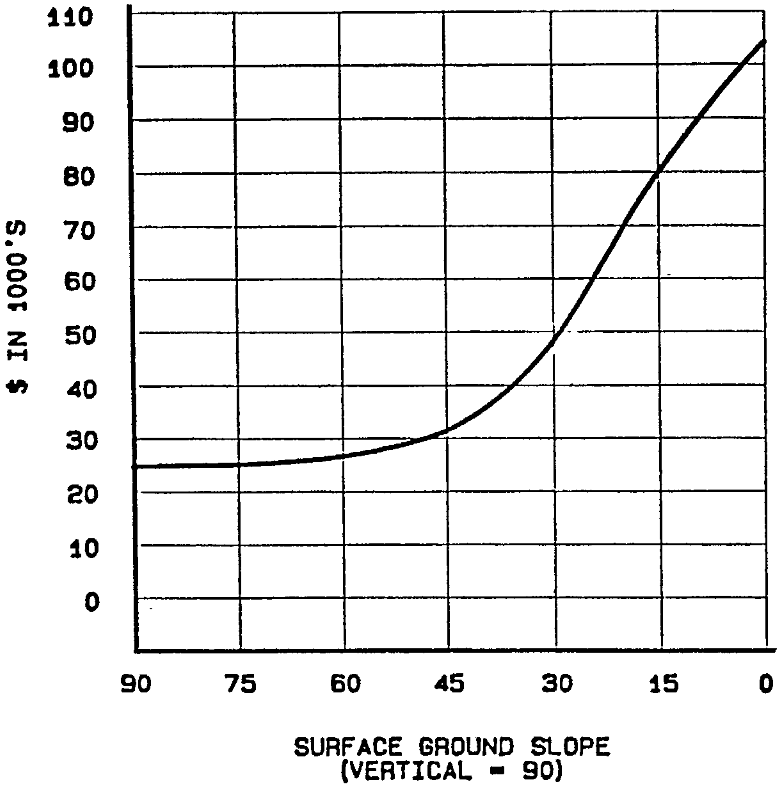
Allow one lump sum of	$160,000

b) Portals in Rock

If the portal is collared directly into rock, the main variable is the ground slope of the area where the portal is excavated.

The graph below indicates costs related to ground slopes ranging from vertical (90°) to flat (0°) .

The costs presented allow for sufficient excavation to establish a 3 metre 'socket' into rock.

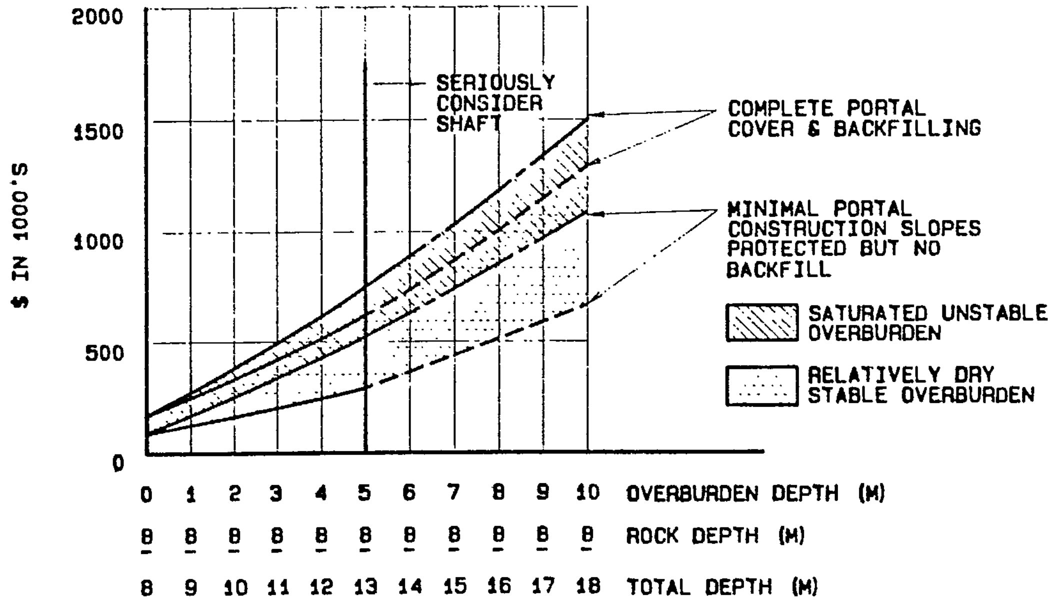


c) Portals in Overburden

If the portal is located in overburden, costs will depend on the depth, type and water content of the overburden.

The costs presented allow for sufficient excavation to establish a 3 metre socket into rock.

Note that the lines are broken after 5 metres of overburden. Should overburden depths in excess of 5 metres be encountered, serious consideration should be given to a shaft.



d) Decline Excavation

Total costs are presented for declines of various gradients.

The choice of decline gradient is a trade-off between capital and operating costs. Steeper grades will require shorter declines but will increase maintenance costs and equipment down-time.

To use the graphs, calculate the depth of the lowest level accessed by the decline and subtract the depth covered by the portal.

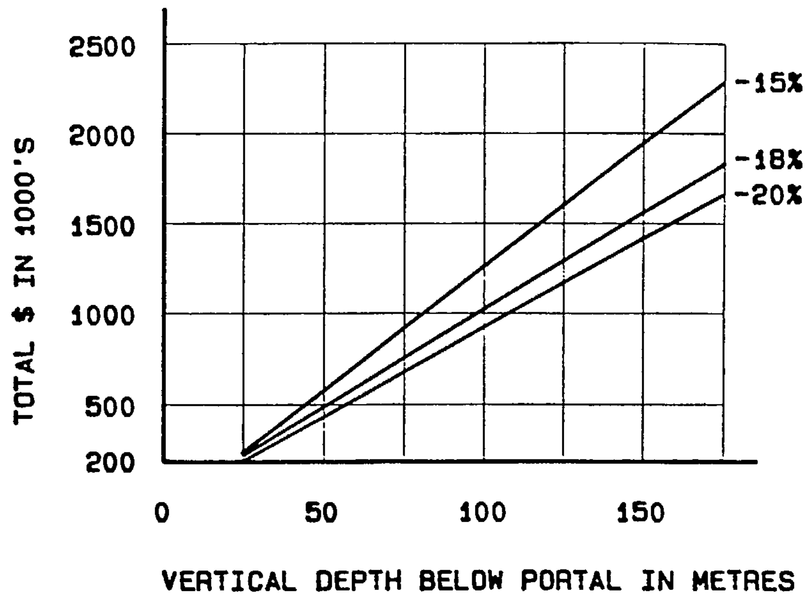
While a low tonnage mine with a life in the order of 5 years is unlikely to recoup the additional cost of a -15% decline, (vs -18% or -20%), some provinces legislate against haulage ramps at steeper grades.

If in doubt, use -15%

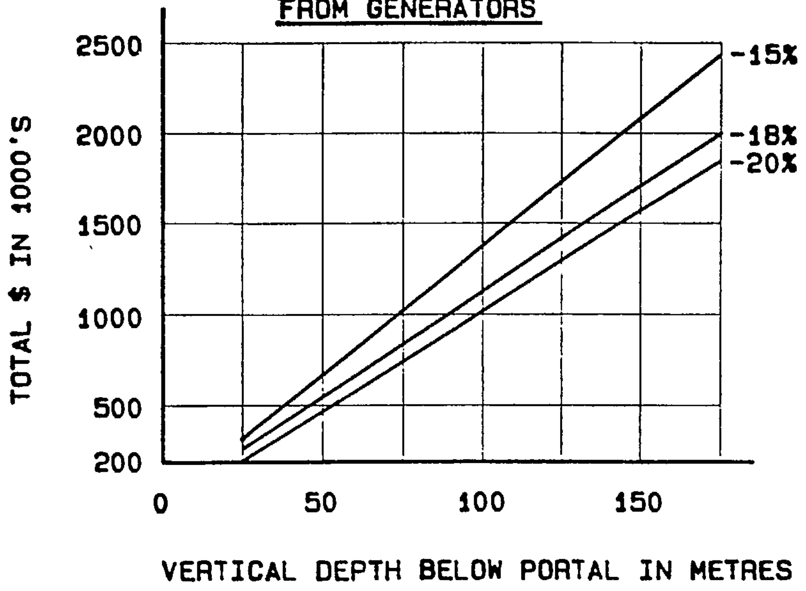
d) Decline Excavation (Continued)

The costs indicated by the graphs below are for a 4.5m x 3.5m decline, fully equipped with air, water and drainage pipelines, ventilation duct and ground support. Quantities used allow for muckbays and miscellaneous excavations.

ELECTRICAL POWER FROM POWERLINE



ELECTRICAL POWER FROM GENERATORS



Note: Remember to allow for vertical depth covered by portal.

3.12.3 Adits

A mine employing an adit or adits as a means of access will fit into one of the following categories:

i) A single adit;
ii) Multiple adits at various elevations;
iii) Either of the above, together with internal ramps. These ramps may be driven upgrade or downgrade.

The user must calculate the length of adit required.

With one adit in place, other levels may be accessed either by additional adits or by internal ramps. If the external terrain permits additional adits the decision will largely depend on the length of adit compared to length of ramp required.

Use the quantities calculated together with the unit costs below to arrive at an estimate of total costs.

a) Mobilization, Setup, Teardown, Demobilize Similar to decline, allow	$160,000

b) Portals

Refer to Section 3.12.2 b), "Portals in Rock".

c) Adits & Ramps - Unit Costs for Excavation

Unit Costs:	$/Metre Electrical Power from Powerline	 Electrical Power from Generators
i) Adits		
Track - 3.5m x 2.5m @ +1%	1400	1550
Trackless - 4.5m x 3.5m @ +1%	1475	1600
ii) Internal Ramps		
Upramps - 4.5m x 3.5m @ +15%	1750	1800
Declines - 4.5m x 3.5m @ -15%	1525	1650

3.13 ANCILLARY SHAFT EXCAVATIONS AND INSTALLATIONS

General

This section covers the excavation, construction and installations required to make the shaft functional.

Included are:

Shaft stations	excavation and construction
Loading pockets	excavation and construction
Lip pockets	excavation and construction
Spill handling	construction
Shaft bottom	construction

3.13.1 Shaft Stations

The number of stations will depend on the overall height of the ore body and the level interval. This will be influenced by the mining method selected. Refer to Section 2.4.

The costs below include excavation, ground support and equipping with station doors, track and pipe.

For 2-compartment timber shafts	$63,500 per station
For 3-compartment timber shafts	$75,000 per station
For 4.5 m diameter concrete shafts	$75,000 per station

3.13.2 Loading Pockets

The loading pocket will be influenced by several factors. These include:

- whether hoisting is carried out in one or two compartments;
- the mine production rate;
- the possibility of future increases in the mine production rate;
- mine life.

Descriptions and costs are provided for four alternatives.

3.13.2 Loading Pockets (Continued)

Costs include excavation, fabrication and installation.

a) Double compartment loading pocket with feed raise, capacity up to 1,000 tonnes per day (ore & waste combined).

$90,000

b) Single compartment loading pocket with feed raise, capacity up to 600 tonnes per day.

$80,000

c) Single compartment loading pocket utilizing two undercutting guillotine gates with feed raise, capacity up to 400 tonnes per day.

$45,000

d) Lip pocket, suitable for production rates under 300 tonnes per day.

$25,000

3.13.3 Lip Pockets (for development tonnages only)

Costs include excavation, materials and installation.

$15,000

3.13.4 Spill Handling

For "simple" arrangement to catch spill at shaft bottom.

$ 5,000

3.13.5 Shaft Bottom Construction

For construction of a small sump and installation of a dirty water pump with basic filtering arrangement.

$10,000

3.14 HOISTING SYSTEMS, HEADFRAMES & BINS

General

This section covers the selection and costing of the components of a hoisting system.

The following are included:

1) hoist and hoistroom;
2) headframe and collarhouse;
3) bin or dump area;
4) conveyances.

Careful consideration should be given to the sizing of a hoisting system. Some 'oversizing' of hoist and headframe will allow for flexibility of operation and allow future increases in production rate to be undertaken at minimal additional expense.

3.14.1 Hoists and Hoistroom

a) Hoist Sizing

The following tables indicate hourly hoisting capacities, for various hoist sizes, hoisting from a depth of 300 metres. There are many other combinations of hoist and skip sizes but the tables should be sufficient to make a suitable selection.

For a three-shift operation, select a hoist that will hoist the required total tonnes of ore and waste in 10 - 14 hours.

i) Using Skip/Cage Combination & Skip

Drum Diameter		Probable Motor Size Range	Payload	Hoisting Capacity
Ft.	(m)	kw	Tonnes	Tonnes/Hour
5	(1.5)	75 - 160	2.0	49 - 78
6	(1.8)	180 - 270	3.0	106 - 128
8	(2.4)	250 - 400	4.0	142 - 170

ii) Using Skip/Cage Combination & Counterweight

Drum Diameter	Probable Motor Size Range	Payload	Hoisting Capacity
Ft. (m)	kw	Tonnes	Tonnes/Hour
5 (1.5)	60 - 100	2.0	25 - 41
6 (1.8)	100 - 160	3.0	55 - 68
8 (2.4)	130 - 225	4.0	74 - 90

b) Costs

The costs presented in the table below include:

- purchase of used hoist
- refurbishing and modification to meet regulations
- hoist installation
- hoisthouse purchase and erection
- foundations and floors

	TOTAL COST		
Hoist Drum Diameter (in feet and metres)	Foundations on Rock	Mat Foundations on Competent Soil	Pile Foundations on Weak Soil
5 ft. (1.52 m)	$500,000	$540,000	$560,000
6 ft. (1.83 m)	$580,000	$620,000	$640,000
8 ft. (2.44 m)	$720,000	$765,000	$790,000

Note:

1) Hoists are used, double drum, with AC motors.

2) Costs assume that hoist is available in dealer's yard - ie. no removal costs.

3) 8-foot (2.4 m) hoists are probably too large for mines under 500 tpd but have been included for:

   i) Possibility of increased production rate;
   ii) Selection due to limited availability of smaller hoists.

3.14.2 Headframes and Collarhouse

a) Design/Selection

Headframes priced in this section are designed for use with 25 mm (1") rope.

Given the rope size, the following decisions are required:

i) Headframe Height - with the limited range of speeds and conveyance sizes probable in mines producing under 500 tonnes per day the major influence on height will be whether or not an ore bin is required and, if so, what capacity.

If a bin is considered necessary, the following size limitations should be taken into account.

Headframe Height	Maximum Bin Capacity	
26 m (85 ft.)	Virtually none	
27.5 m (90 ft.)	100 tonne	for truck
30.5 m (100 ft.)	200 tonne	loading
33.5 m (110 ft.)	300 tonne	

ii) Steel or Timber Construction

No significant cost difference.

Eventual decision will likely depend on delivery schedule, transportation cost and personal preference.

iii) Moving an Existing Headframe - often not as economic as might be expected.

Two exceptions are:

- if the headframe was designed to be portable.
- if property location makes transportation charges excessive and existing headframe is close.

b) Costs

The costs presented below include:

- purchase of new headframe
- headframe erection
- collarhouse purchase and erection
- headframe cladding
- headframe equipping
- headframe & collarhouse footings & floors.

i) New Headframes

Headframe Height	$
26 m (85 ft.)	265,000
27.5 m (90 ft.)	285,000
30.5 m (100 ft.)	315,000
33.5 m (110 ft.)	350,000

ii) Moving Existing Headframes

As an approximation use 80% of equivalent new headframe cost.

Note:

1) Headframes suitable for use with 25 mm (1") rope.

2) Collarhouse assumed to be a new 9 m x 9 m pre-engineered building with floor and track.

3) Headframe equipped with guides, limits, shaft doors, sheaves, and brattice.

4) Headframe cladding is single skin - no insulation.

3.14.3 Headframe Bins

a) Selection

Because of the high cost of a bin of any appreciable size, the requirement for a bin should be given serious thought.

Bins offer advantages in loading either trucks or conveyors but can experience problems with freezing, particularly if not heated.

Should a bin be considered necessary, note the capacity limitations versus headframe height listed in Section 3.14.2.

b) Costs

Costs presented below include:

- purchase and delivery of new steel bin
- footings
- erection
- simple unheated enclosure

	$
i) 100 tonne	125,000
ii) 200 tonne	200,000
iii) 300 tonne	275,000

c) Bin Alternative

As an alternative to a headframe bin, hoisted muck can be dumped onto the ground to be picked up by a loader.

For the construction of a timber and concrete "dump area", allow $ 15,000

3.14.4 Conveyances

Probable conveyance combinations are as follows:

- Two-compartment shafts - skip/cage combination & counterweight
- Three-compartment shafts - skip/cage combination & skip
  - separate skip & cage

Prices are presented below for new skips of various capacities and cages with capacity of 12 to 13 men.

	$		
	2-Tonne Skips	3-Tonne Skips	4-Tonne Skips
i) Skip/cage combination & cwt.	72,000	76,000	82,000
ii) Skip/cage combination & skip	85,000	93,000	100,000
iii) Skip & cage (separate units)	60,000	65,000	70,000

Note:

These are purchase and delivery costs only. Suspending conveyances in shaft is covered elsewhere.

3.15 VENTILATION AND MINE AIR HEATING

General

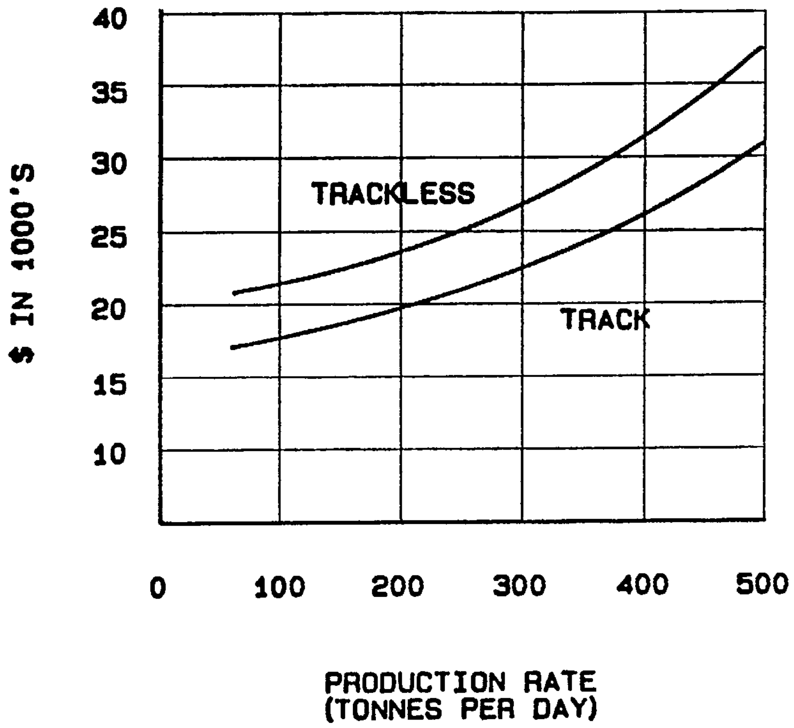
This section covers the supply and installation of new:

i) Primary ventilation fans.
ii) Mine air heaters.

3.15.1 Primary Ventilation Fans

Ventilation air volumes and resistances have been estimated for a typical case mine. This information has been used to select and cost the appropriate fans.

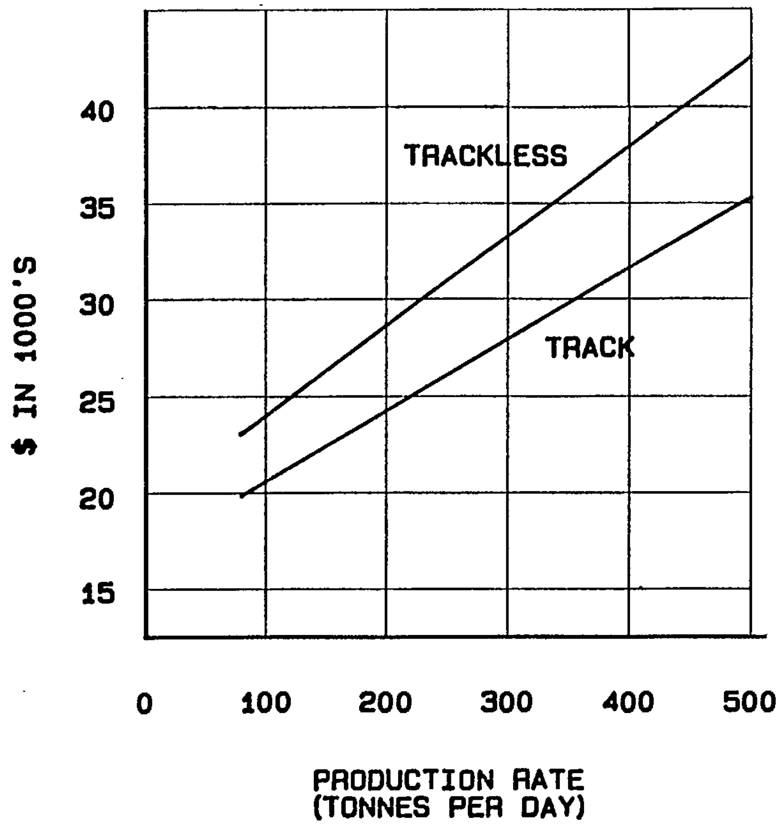
Select ventilation fan capital costs from the graph below based on the mine production rate and type of equipment used underground.



3.15.2 Mine Air Heaters

The mine air heating plant is based on using direct-fired propane heating and is sized and costed according to the ventilation criteria previously established.

Select a capital cost for mine air heaters from the graph below based on the mine production rate and type of equipment used underground.



3.16 UNDERGROUND DEVELOPMENT

General

This section covers the following:

a) Level development (crosscuts and drifts in waste), from shaft, ramp or adit prior to the start of production. Development directly related to stoping is not included as it is covered in operating costs.

b) Raise development to establish an ore pass system (if required).

c) Raise development and construction to establish a primary ventilation and escapeway system.

A sketch is located following section 3.16.3 to clarify some of the terms used.

A summary of unit rates for excavation is included in Appendix 3.B.

3.16.1 Level Development

The preproduction capital costs calculated in this section allow for the completion of sufficient development to access stoping blocks containing tonnages equivalent to two years production.

Capital development beyond this amount is covered under "Ongoing Capital Costs".

The quantity and cost of preproduction development (P.P.D.) will depend on the following variables.

Variables:
Annual production rate
Strike Length (within mining limits)
Average stoping width
Density of material to be mined
Interval between main levels
Average crosscut length

A method of estimating preproduction development (P.P.D.) costs is outlined below. The calculation is repeated in the capital cost calculation form for the user to enter the appropriate information.

P.P.D. costs are approximated by the following:

$$\text{P.P.D. COSTS} = \frac{\text{QTY. OF DEV'T/LEVEL(m) x COST/METRE x TONNES/YR. x 2}}{\text{TONNES ACCESSED/LEVEL}}$$

In order to define the terms used above, remembering that '2' accounts for the 2 years of development required, this expression is rewritten as:

$$\text{P.P.D. COSTS} = \frac{(a) \times (b) \times (c) \times 2}{(d)}$$

(a) = quantity of development per level
= average crosscut length (m) + strike length (m)

(b) = cost per metre

	Shaft Access Track Drift 2.4 m x 3.0 m	Shaft Access Trackless Drift 4.0 m x 3.0 m	Ramp or Adit Access Trackless Drift 4.0 m x 3.0 m
Power from utility	$1390	$1440	$1475
Power from generators	$1525	$1570	$1600

(c) = tonnes per year

= working days per year x daily production (tonnes)

(d) = tonnes accessed per level calculated as follows:

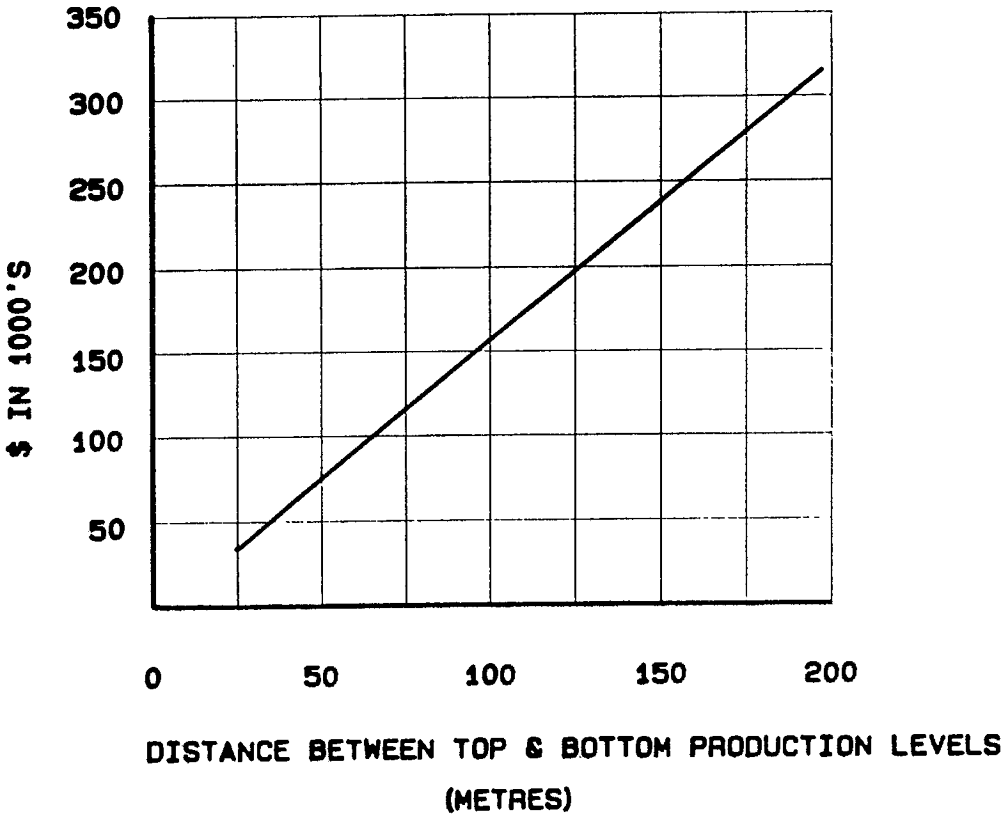
Strike Length (m) x Average Stoping Width (m) x Ore Length* Between Levels (m) x Density of Ore (tonnes/m^3)

* Measured along dip. For vertical ore bodies this becomes the level interval.

3.16.2 Ore Pass System

The costs indicated on the following graph were calculated based on the parameters detailed below:

- ore pass extends between top and bottom production levels.
- raise size is 2.13 m x 2.13 m
- raise angle is 70°
- level interval is 50 m with a 10 m finger raise at each level except the top level.
- costs include excavation and ground support. No construction is included. Ore pass controls and grizzlies are covered in Section 3.17.



If the user has identified the quantities of raise development required, the unit rate below can be used to estimate costs.

Cost per metre (2.13 m x 2.13 m) - $1,320

Note: If production is from one level only, or if ore is trucked directly to surface, an ore pass system may not be required.

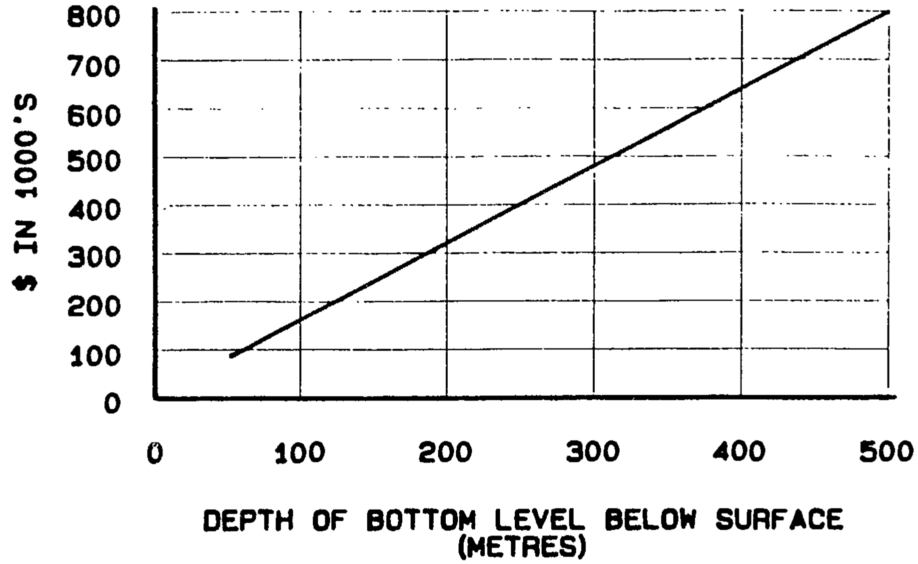
It is assumed that a separate waste pass is not required. Preproduction waste would be handled through the ore pass system (once available) and by lip pockets on the levels once production starts.

Should a waste pass system be considered necessary, costs would be similar to the ore pass system.

3.16.3 Primary Ventilation and Escapeway

The costs indicated on the following graph were calculated based on the parameters detailed below:

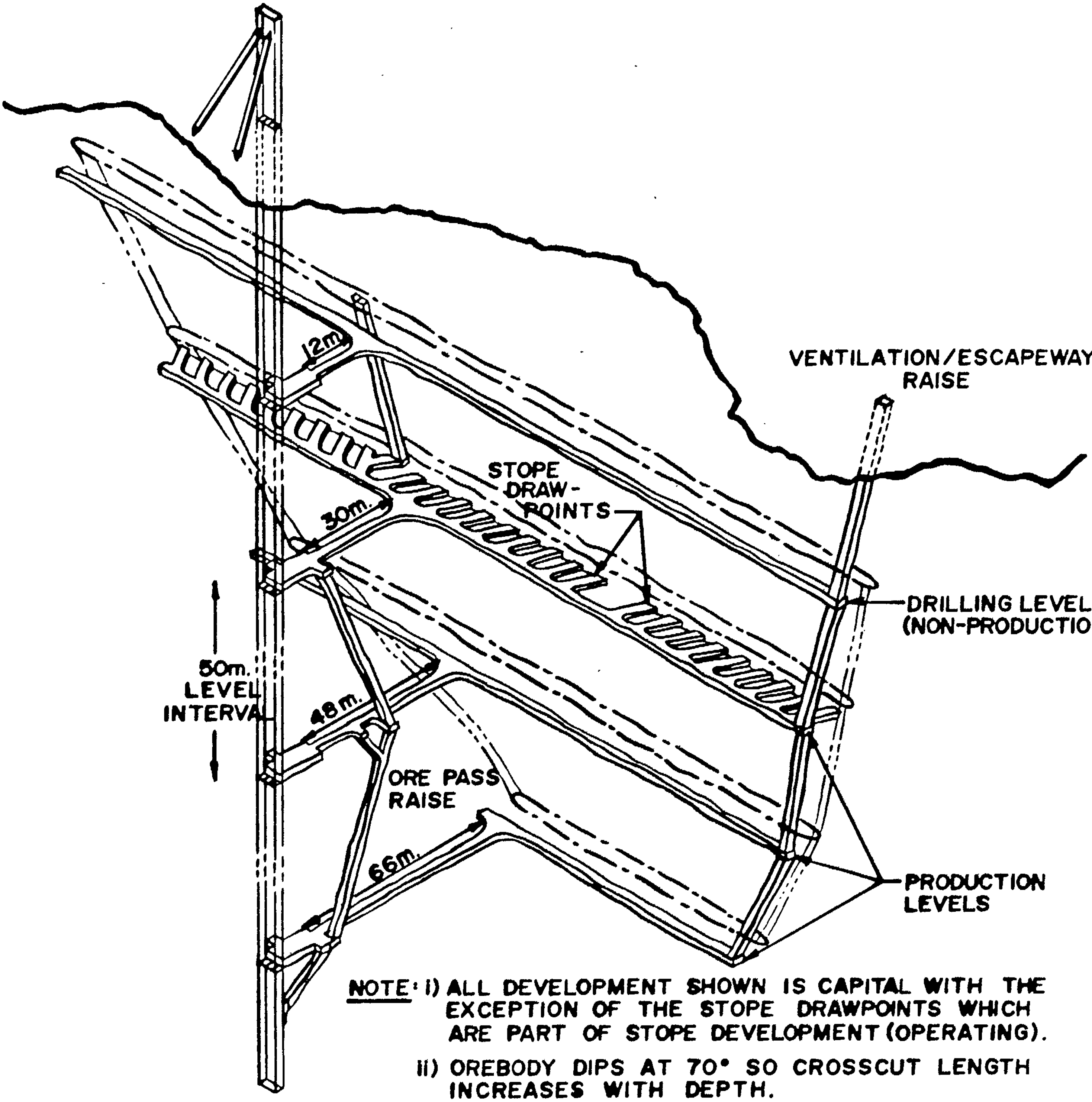
- the raise extends from bottom production level to surface.
- raise size is 1.83 m x 2.44 m (alimak).
- raise angle is 80°.
- the raise includes a timber manway.
- costs include excavation, ground support and manway installation.
- the breakthrough to surface is assumed to cause no significant problems.



If the user has identified the quantity of raise development required the unit rate below can be used to estimate costs.

Cost per metre (1.83 m x 2.44 m) - $1,650

This unit rate includes setup, teardown, ground support and the cost of alimak cut-outs.



NOTE: I) ALL DEVELOPMENT SHOWN IS CAPITAL WITH THE EXCEPTION OF THE STOPE DRAWPOINTS WHICH ARE PART OF STOPE DEVELOPMENT (OPERATING).

II) OREBODY DIPS AT 70° SO CROSSCUT LENGTH INCREASES WITH DEPTH.

CUT-AWAY VIEW OF TYPICAL MINE LAYOUT

3.17 UNDERGROUND INSTALLATIONS

General

This section covers underground installations, including any additional excavation that might be required, for items such as:

- Main sumps and pump stations.
- Rockbreaker and grizzly.
- Ore pass controls.
- Underground electrical room.
- Other miscellaneous installations.

3.17.1 Main Sumps and Pump Stations

Costs will depend on water inflow, mine depth and whether mine is accessed by ramp or shaft.

Costs include the supply and installation of pumps and the excavation and construction required for clear and dirty water sumps.

	Mine Layout		
	Ramp to 150 m depth	Shaft to 200 m depth	Shaft to 400 m depth
'Dry' Mines	$45,000	$50,000	$ 65,000
'Average' Mines	65,000	72,000	80,000
'Wet' Mines	80,000	90,000	100,000

3.17.2 Rockbreaker and Grizzly

Allow one rockbreaker and grizzly installation in shaft mines to condition muck prior to loading skip.

For mines that use ramp haulage, no rockbreaker required.

Costs include the supply and installation of a new pneumatic rockbreaker, the construction of a grizzly and the additional excavation required for each.

Cost per installation	$95,000

3.17.3 Ore Pass Controls

Allow a set of control chains every second level.

Costs include the supply and installation of a brow beam and chains c/w cylinder and controls.

Cost per control	$20,000

3.17.4 Underground Electrical Room/Load Centre

To provide working voltages underground, allow one substation every second level.

The cost presented represents the supply and installation of a 200 KVA substation.

Cost per substation	$37,000

3.17.5 Miscellaneous Installations

To allow for the numerous smaller construction items required underground a "miscellaneous" allowance per level is provided.

Allowance per level	$25,000

3.18 EQUIPMENT

General

This section covers the cost of equipment required for production mining and support services.

Depending on the mining layout this equipment may include:

- trackless loading and haulage equipment.
- track loading and haulage equipment.
- development and production drills.
- secondary fans and pumps.
- underground service vehicles.
- surface payloader.

It does not include mine plant such as hoists, compressors, generators or main mine pumps which are covered individually elsewhere.

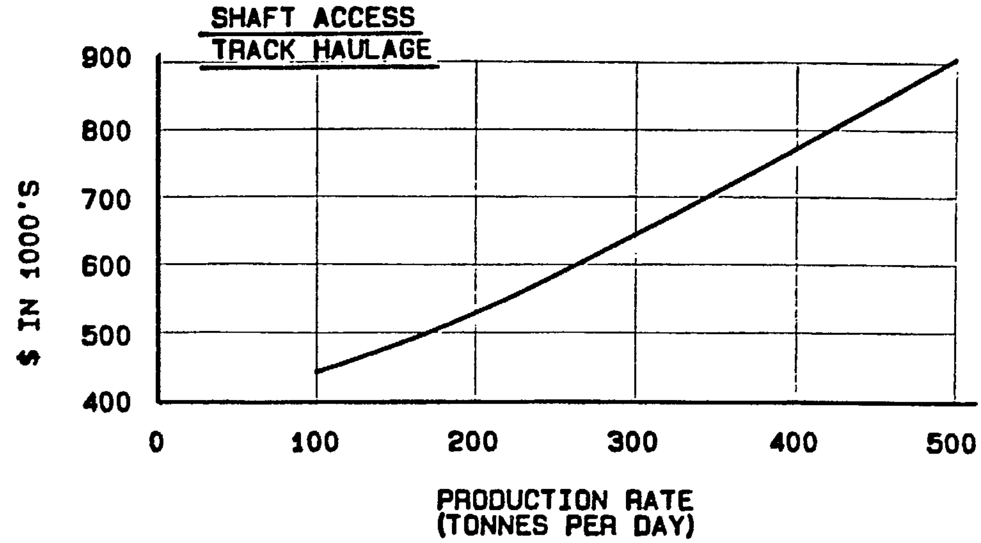
For simplicity, it is assumed that equipment is purchased outright. Should this cause initial capital costs to become excessively high, then a portion of the equipment could be considered to be acquired on a rental purchase basis. Any lowering of capital costs due to renting equipment, however, would require an offsetting increase in the operating cost per tonne.

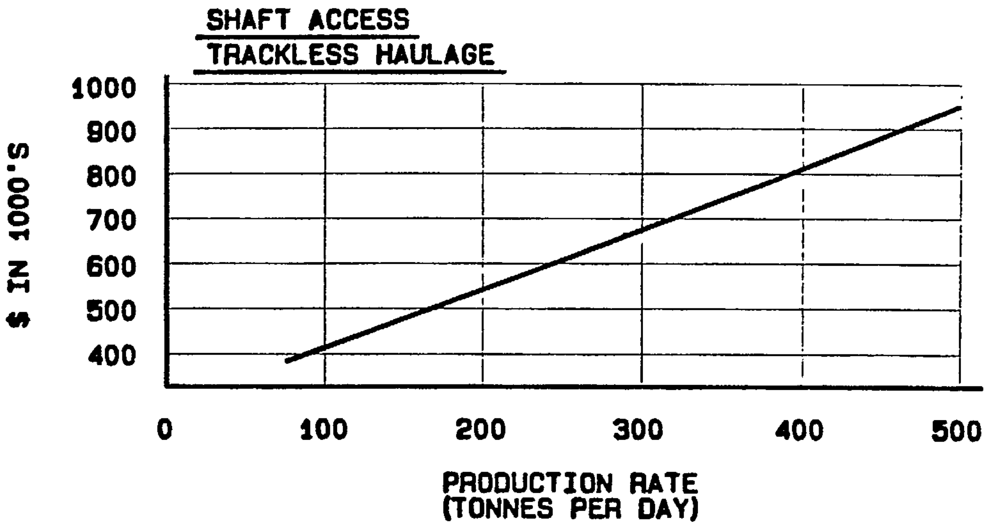
Costs

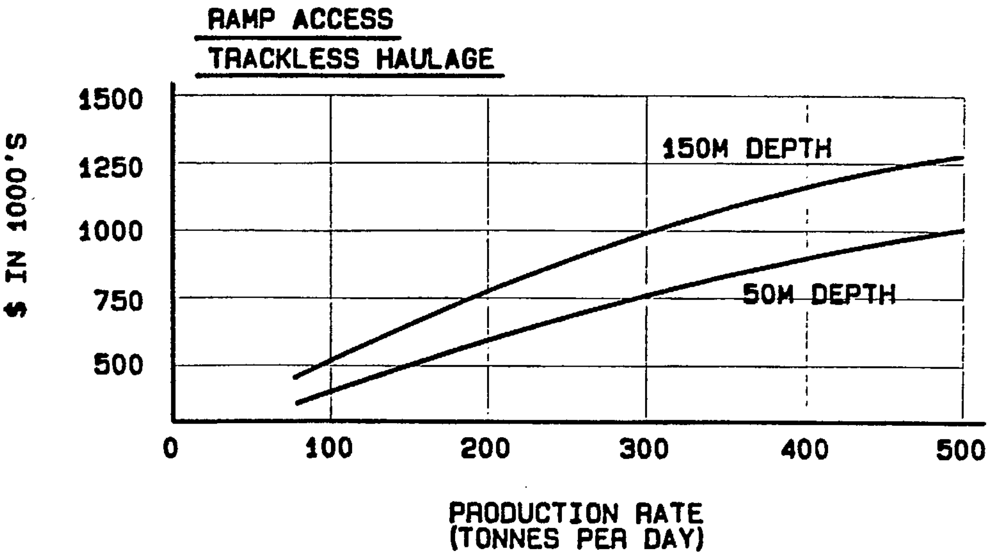
The costs presented on the following graphs are based on quantities and equipment types considered "typical" of mines producing at tonnages of 100 to 500 tonnes per day. An allowance for a spares inventory has also been included.

Wherever practicable, "good-used" values for equipment have been used. For example, all haulage equipment including L.H.D. units, trucks, mucking machines, locomotives and mine cars have been included at used prices. All production drilling equipment is priced new.

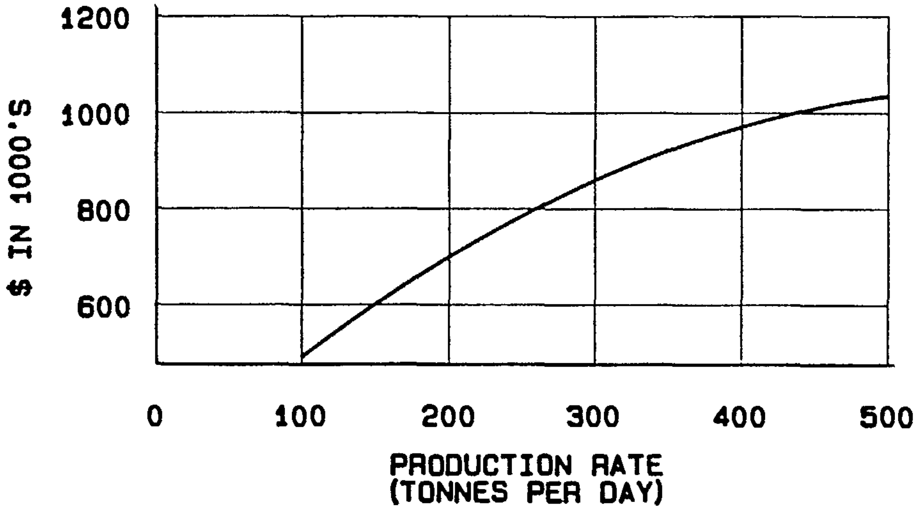
Note: New and "good-used" values of a variety of commonly used mining equipment are included in Appendix 3.A.


SHAFT ACCESS
TRACK HAULAGE
$ IN 1000'S
900
800
700
600
500
400
0
100
200
300
400
500
PRODUCTION RATE
(TONNES PER DAY)

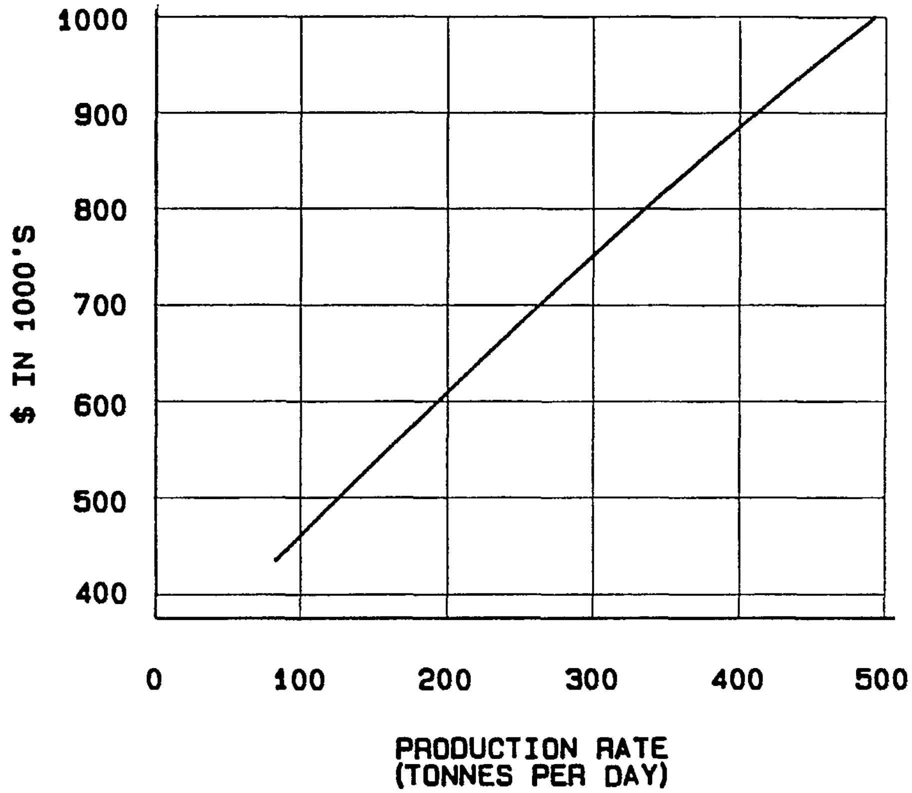

SHAFT ACCESS
TRACKLESS HAULAGE
$ IN 1000'S
1000
900
800
700
600
500
400
0
100
200
300
400
500
PRODUCTION RATE
(TONNES PER DAY)


RAMP ACCESS
TRACKLESS HAULAGE
$ IN 1000'S
1500
1250
1000
750
500
150M DEPTH
50M DEPTH
0
100
200
300
400
500
PRODUCTION RATE
(TONNES PER DAY)

ADIT ACCESS
TRACKLESS HAULAGE



ADIT ACCESS
TRACK HAULAGE



3.19 CONCENTRATOR

General

This section covers the cost of constructing a concentrator where the possibility of shipping to an existing concentrator is not feasible or is contraindicated due to other considerations.

The design and cost of a mineral concentrator is dependent upon a number of variables including, but not limited to:

- Production rate.
- Type of mineral.
- Mineral association.
- Rock hardness.
- Comminution required to liberate mineral constituents.
- Concentrating process.
- Contaminant minerals.
- Tailings disposal.
- Flowsheet.

3.19.1 Concentrator Construction

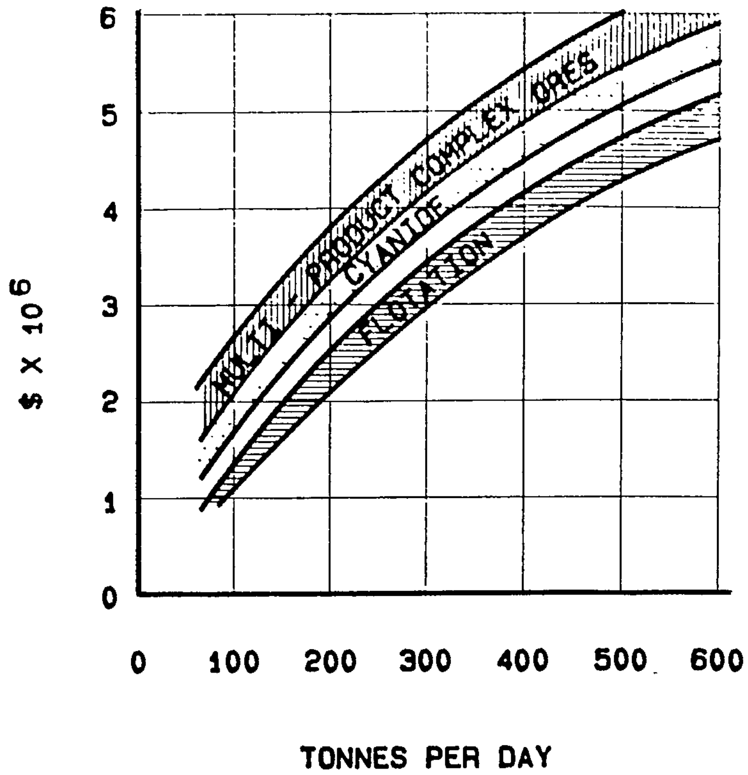
The first graph on the following page gives a range of typical costs for concentrators treating various types of ores. Flotation would be used in a base metal operation whereas cyanidation would be used in a gold mining scenario.

Multi-product or complex ores are those ores requiring special treatment due to the complex character of the ore which necessitates finer grinding and intensive treatment, or ores which require a combination of treatment methods.

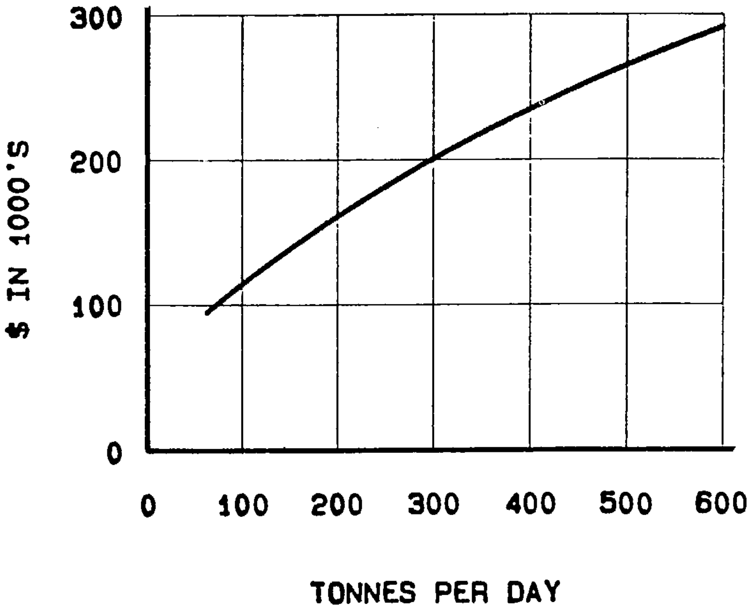
3.19.2 Tailings Disposal Area

The second graph on the following page indicates the cost of constructing a tailings disposal area based on the mine production rate.

CONCENTRATOR CONSTRUCTION



TAILINGS DISPOSAL AREA



3.20 COST CONTINGENCY

General

The cost contingency estimated in this section is intended to cover conditions beyond the control of the operator, and is not meant to compensate for poor or incomplete estimating.

The costs allocated to contingency are expressed as percentages of overall preproduction capital costs and are divided into three major areas as follows:

i) The items which may have been omitted from the manual. The contingency allocated is intended to cover numerous small items rather than individual items of significant cost.

ii) Variations in conditions actually encountered from those anticipated.

iii) Delays influenced by project location.

Review the sections below and compile an overall contingency percentage accordingly.

3.20.1 Items Omitted

Although every effort has been made to include all identifiable costs, it is certain that some minor items have been omitted.

For these additional items allow 5%

3.20.2 Variations in Conditions

This estimate of costs is likely being prepared at a time when the availability of definite information regarding the project is limited. This fact necessitates that the user make a number of decisions based on "best judgement". The accuracy of these decisions will have a significant impact on the overall accuracy of the estimate.

The allocation of contingency requires the user to review those decisions and assess the risk involved in the decisions made.

For example, one person may have assumed the "worst case" each time a judgement was required, and therefore produced a very conservative estimate of costs.

Conversely, a second person with identical circumstances may have made "hopeful" or "best case" decisions and therefore produced a very optimistic estimate of costs.

Clearly, a different contingency would be appropriate in each case.

To determine a contingency percentage, review the major decisions you have made and make an assessment of the potential for increases in costs should you be incorrect. Then select a percentage from the range offered below:

Assessment of Decisions	Contingency Percentage
Very conservative	0%
Relatively optimistic but confident decisions are correct	5%
Optimistic with low level of confidence in decisions	10%

3.20.3 Delays

All projects are subject to delays due to a variety of reasons. These reasons are numerous but include:

i) Local weather conditions;

ii) Delays in scheduled and unscheduled deliveries to the project site. This problem can be compounded if bad weather is also encountered.

iii) The requirement of specialized services, material or equipment not immediately available at or near the site.

The location and/or remoteness of a project will affect all of the above.

Bearing in mind that costs presented in this manual are based on the climate, transportation system and availability of specialized services, material and equipment found in north-central Ontario, assess the affect of delays due to the location of the project being evaluated.

Estimate within a range of 0 - 5%.

Summary

After reviewing the three sections above, the user should total the three contingency items to determine the overall contingency which will be in the range of 5 - 20%.

3.21 ONGOING CAPITAL DEVELOPMENT

General

This section covers the cost of development required to provide access to new stoping blocks as working stopes are mined out.

A means of determining the daily manpower required to complete this development is outlined on the following page.

Costs

Costs are developed on the basis that sufficient development is completed annually to provide access to new stoping blocks containing tonnages equivalent to one year's production.

The calculation below is identical to that used to calculate preproduction capital costs for level development (Section 3.16.1) with three exceptions:

i) ongoing capital development (O.C.D.) costs allow for development of <u>one</u> year's tonnage instead of two.

ii) the cost per metre allows for the Owner's direct labour, consumables and direct operating supplies only.

iii) Annual O.C.D. costs are approximated by the following:

$$\text{ANNUAL O.C.D. COSTS} = \frac{\text{QTY. OF DEV'T/LEVEL (m)} \times \text{COST/METRE} \times \text{TONNES/YR.}}{\text{TONNES ACCESSED/LEVEL}}$$

Owner's cost per metre:

		$/metre
Track drift (2.44 m x 3.05 m)	- shaft access	800
Trackless drift (3.0 m x 4.0 m)	- shaft access	800
	- ramp access	950

Note: The total amount of O.C.D. can be determined by calculating how many levels, or what portion of a level, is required to provide one year's production.

Manpower

The total annual quantity of "Ongoing Capital Development" can be determined by dividing the total annual cost by the cost per metre.

That is,

$$\text{ANNUAL QTY. OF O.C.D.} = \frac{\text{ANNUAL O.C.D. COST}}{\text{COST/METRE}}$$

With the annual quantity of O.C.D. established, the average daily manpower required can be approximated by:

$$\text{AVERAGE DAILY MANPOWER} = \frac{\text{ANNUAL QUANTITY OF O.C.D.}}{\text{PERFORMANCE/MANSHIFT} \times \text{WORKING DAYS/YEAR}}$$

For typical performances use:

- track 0.75 metres/manshift
- trackless 1.0 metres/manshift

Round up the resulting figure to the next whole number.

3.22 EXPLORATION DEVELOPMENT

General

This section covers the cost of development carried out solely for exploration purposes. This development may, or may not increase the mineral inventory.

Costs

The amount of exploration development required annually is not a calculable quantity. It will vary from year to year with the availability of funds and the urgency of increasing the mineral inventory.

It is fair to say that exploration is related, in the long run, to annual production rate and, therefore, ongoing capital development.

Allow 20% of the ongoing capital development cost calculated in Section 3.21.

3.23 EXPLORATION DIAMOND DRILLING

General

This section covers the cost of diamond drilling carried out in an attempt to increase the mineral inventory.

Assuming that drilling is carried out as part of a regular program the annual exploration diamond drilling (E.D.D.) cost will depend on:

i) Quantity of drilling per setup;

ii) Setup interval along strike;

iii) Metres of exploration development completed annually;

iv) Cost per metre drilled.

Costs

$$\text{ANNUAL E.D.D. COST} = \text{QTY. DRILLING/SETUP} \times \frac{\text{ANNUAL QTY. OF EXPLORATION DEV'T}}{\text{SETUP INTERVAL}} \times \text{COST/METRE DRILLED}$$

$$= \frac{(a) \times (b) \times (c)}{(d)}$$

(a) = quantity of diamond drilling per setup in metres.

Allow four holes per setup. Average length to be determined by the user depending on the geometry of the deposit.

If in doubt, use four holes at 60 metres each.

(b) = annual quantity of exploration development in metres.

$$= \frac{\text{Annual O.C.D. Costs (Section 3.21) x 20\%}}{\text{Cost/Metre (Section 3.16.1)}}$$

(c) = cost per metre for diamond drilling (Section 3.3.2)

(d) = setup interval

Use a setup interval along strike of 25 metres.

3.24 EQUIPMENT REPLACEMENT

General

This section covers the ongoing replacement of major equipment necessary because of depreciation or damage.

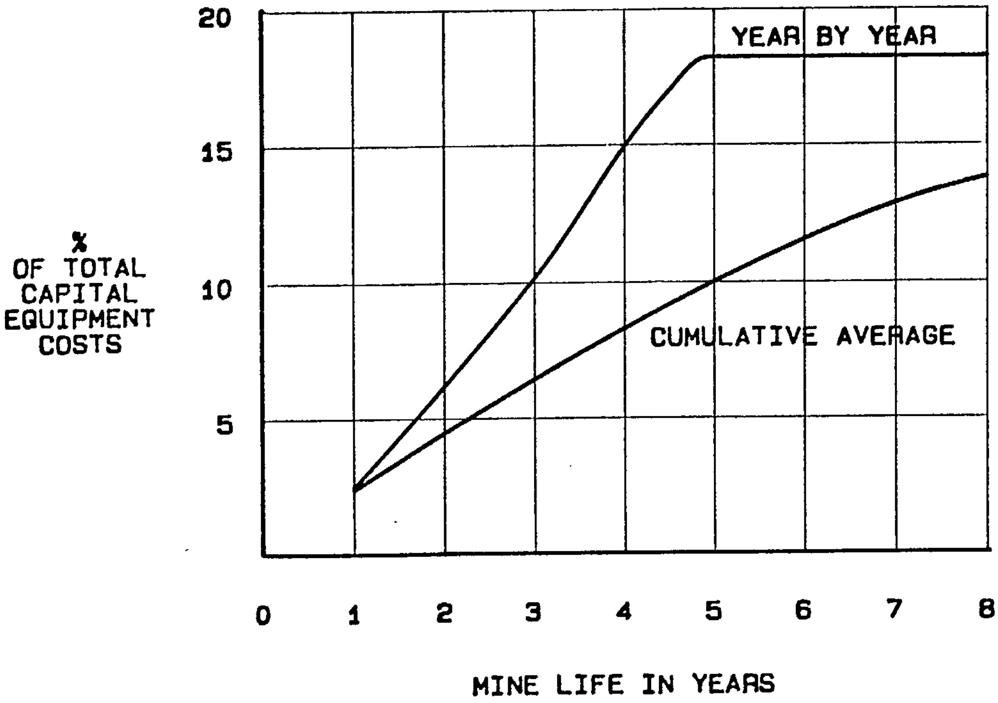
Costs

Equipment replacement costs will vary depending on the following:

i) Total value of equipment on site.
ii) Years of use since purchase.
iii) Whether new or used equipment was purchased initially.

The graph below approximates the annual replacement cost of major equipment. It expresses costs, by year, as a percentage of total capital equipment costs as determined in Section 3.18.

Both year by year and cumulative average percentages are indicated. An example calculation can be found in Appendix A, Form 3(b), page 3 of 3.



APPENDIX 3.A

EQUIPMENT CAPITAL COSTS

If equipment requirements have already been detailed by the user, the following list will provide costs for a range of commonly used equipment.

Costs are F.O.B. the manufacturer's plant or sellers' yard, and do not include taxes.

Item	Size or Type	New $	Good Used $
L.H.D. Units	1.0 yd.^{3}	77,500	55,000
	2.0 yd.^{3}	125,000	80,000
	2.2 yd.^{3}	131,000	85,000
	2.5 yd.^{3}	138,000	90,000
	3.5 yd.^{3}	141,000	105,000
	5.0 yd.^{3}	212,000	135,000
	6.0 yd.^{3}	219,000	150,000
U/G Trucks	13-ton	145,000	90,000
	15-ton	147,000	90,000
	26-ton	240,000	125,000
Drill Jumbos	2-boom pneumatic	190,000	145,000
	3-boom pneumatic	245,000	170,000
	1-boom hydraulic	250,000	135,000
	2-boom hydraulic	350 - 425,000	250,000
Production Drills	I.T.H. drill c/w 40m Rods	75,000	45,000
	Fan drill (upholes) - pneumatic	160,000	100,000
	Fan drill (360°) - hydraulic	260,000	160,000
	Longhole wagon (35 - 64 mm)	65,000	40,000
	Bar and arm	25,000	16,000

Item	Size or Type	New $	Good Used $
Handheld Drills	Jackleg	2,400	1,500
	Stoper	2,400	1,500
	Plugger	2,200	1,300
Service Vehicles	Scissorlift	75,000	50,000
	Personnel Carrier (4 man)	17,000 - 25,000	10,000
Bulldozers	D-3	74,000	50,000
	D-4H	88,000	60,000
	D-5H	149,000	80,000
Track Mucking Machines	0.26 m^3	52,870	20,000
	0.40 m^3	91,000	45,000
	0.60 m^3	125,000	60,000
Underground Battery Locos	1½-Ton	34,000	12,000
	3½-Ton	45,000	22,000
Mine Cars	2-Tonne	4,200	1,300
	4-Tonne	10,000	3,750
Payloaders	915	98,000	75,000
	926	109,000	90,000
	936	141,000	110,000
Compressors	0.35 m^3/sec. (750 cfm)	40,000	25,000
	0.71 m^3/sec. (1500 cfm)	70,000	45,000
Generators	100 kW	27,000	15,000
	400 kW	65,000	45,000
	1000 kW	125,000	75,000

APPENDIX 3.B

UNIT RATES FOR UNDERGROUND DEVELOPMENT

For the user who has detailed underground layouts established, unit rates for various underground excavations including shafts, track and trackless drifts, raises (alimak, conventional and bored), etc. are listed below.

Rates are all-inclusive and assume Contractor's productivity.

Item	Size or Type	$/Metre Hydro Power	$/Metre Generated Power
Shafts	2-Comp't. timber	4,500	5,000
(complete with bearing	3-Comp't. timber	5,500	6,100
sets, catch pits, etc.)	4.5 m dia. concrete	7,500	8,200
Access Ramps	4.5 m x 3.5 m (-15%)	1,700	1,850
Adits	3.5 m x 2.5 m track	1,400	1,550
	4.5 m x 3.5 m trackless	1,475	1,600
Drifts - Track (shaft access)	2.4 m x 3.0 m	1,390	1,525
Drifts - Trackless (shaft access)	3.0 m x 4.0 m	1,440	1,570
- Trackless (ramp access)	3.0 m x 4.0 m	1,475	1,600
Raises - Open	1.83 m x 1.83 m	975	1,025
- Timber	1.52 m x 2.0 m	1,425	1,500
- Alimak (cut-out separate)	2.0 m x 2.0 m	1,200	1,245
- Raisebore (station separate)	1.22 m dia.	800	900
	1.52 m dia.	900	1,025
	1.83 m dia.	1,000	1,150
Miscellaneous Excavations	Large or simple excavation	76/m^3	84/m^3
	Small or difficult excavations	180/m^3	200/m^3

SECTION 4

4.0 REGIONAL COST FACTORS

4.0 REGIONAL COST FACTORS

General

Capital and mine operating costs are significantly influenced by the geographical location of a project.

The cost information contained in this manual has been developed based on conditions prevailing in north-central Ontario. This "base region" includes the mining camps of Elliot Lake, Sudbury and Timmins.

The purpose of this section is to determine the regional cost factors which must be applied to the capital and operating costs presented in the manual to adjust these costs for specific project locations.

4.1 BASIC APPROACH TO DETERMINING COST FACTORS

Individual factors have been derived, in each region, for the following cost components.

i) Labour
ii) Plant and Equipment
iii) Materials and Consumables
iv) Hydro Power
v) Transportation
vi) Provincial Tax

These individual cost factors are tabulated in Section 4.3.

By estimating the percentage of each component in either operating or capital costs and applying the appropriate cost factor to each, an overall cost factor can be obtained.

Examples are included in Section 4.4.

The user can determine the appropriate cost factors in two ways:

i) By referring to the two maps in Section 4.2 a blanket factor can be selected. These blanket factors have been calulated based on parameters described in Section 4.2.

ii) By breaking down the estimates of capital and operating costs (separately) into percentages of each of the six cost components and then applying the appropriate individual factors, to determine an overall factor that is specific to the user's project. Use Form 4 to calculate these factors.

4.2 BLANKET COST FACTORS (MAPS)

Blanket cost factors are indicated on the maps, Figures 4 - 1 and 4 - 2, for capital and operating costs respectively. The factors indicated are based on the following scenarios:

Capital Cost Criteria:

1) 200 tonne per day operation.
2) On site mill.
3) Decline access.
4) Road access to minesite.
5) Power generated on site.

Item	Distribution of Capital Cost
Labour	40%
Plant and Equipment	20%
Materials & Consumables	35%
Hydro Power	0%
Transportation	3%
Provincial Tax	2%

Operating Cost Criteria:

1) 200 tonne per day operation.
2) Blasthole stoping.
3) Decline access.
4) On-site milling.
5) Power generated on site.

Item	Distribution of Operating Cost
Labour	54%
Plant and Equipment	0%
Materials & Consumables	42%
Hydro Power	0%
Transportation	3%
Provincial Tax	1%

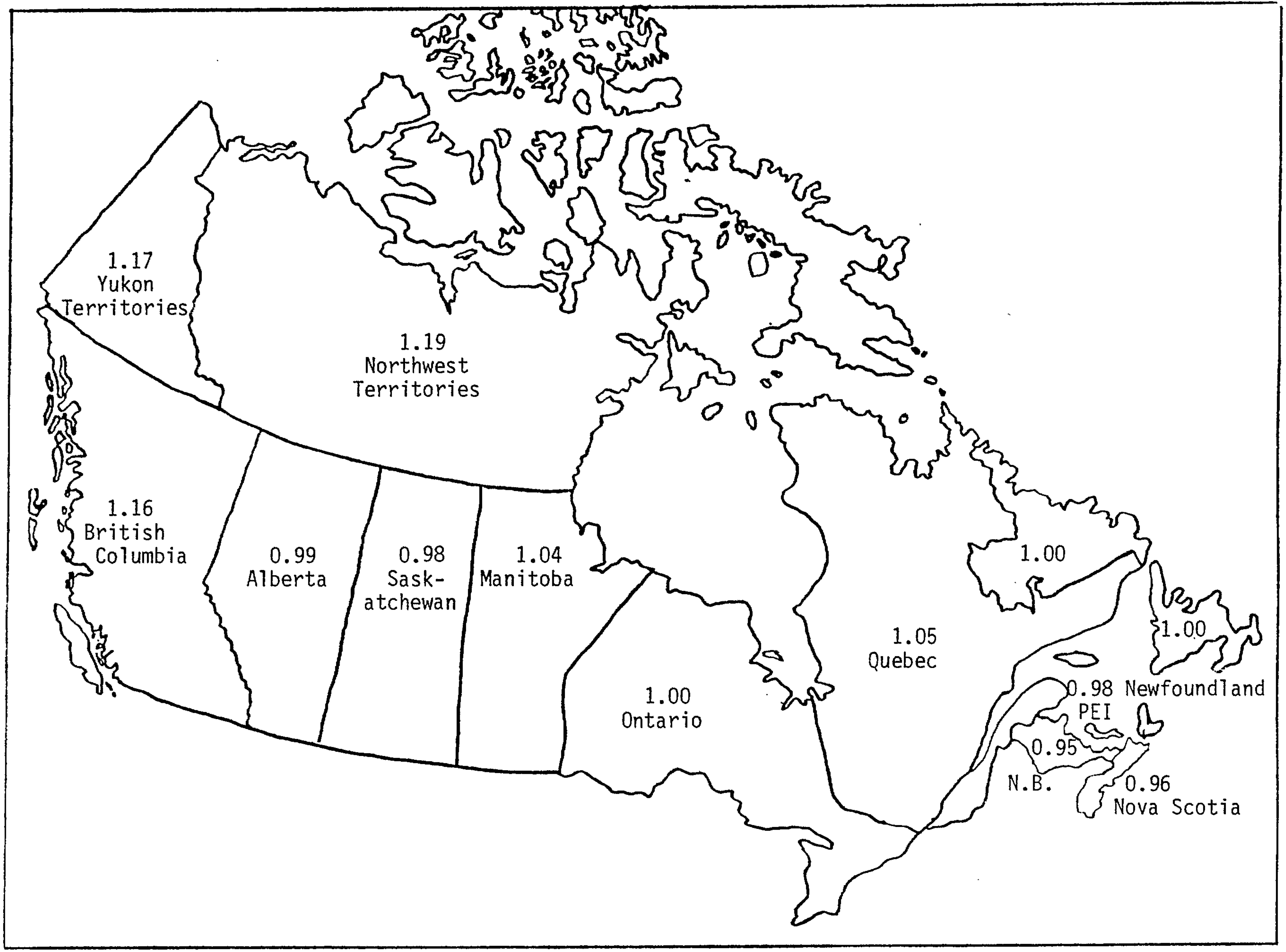


FIGURE 4 - 1

CAPITAL COSTS
BLANKET REGIONAL COST FACTORS

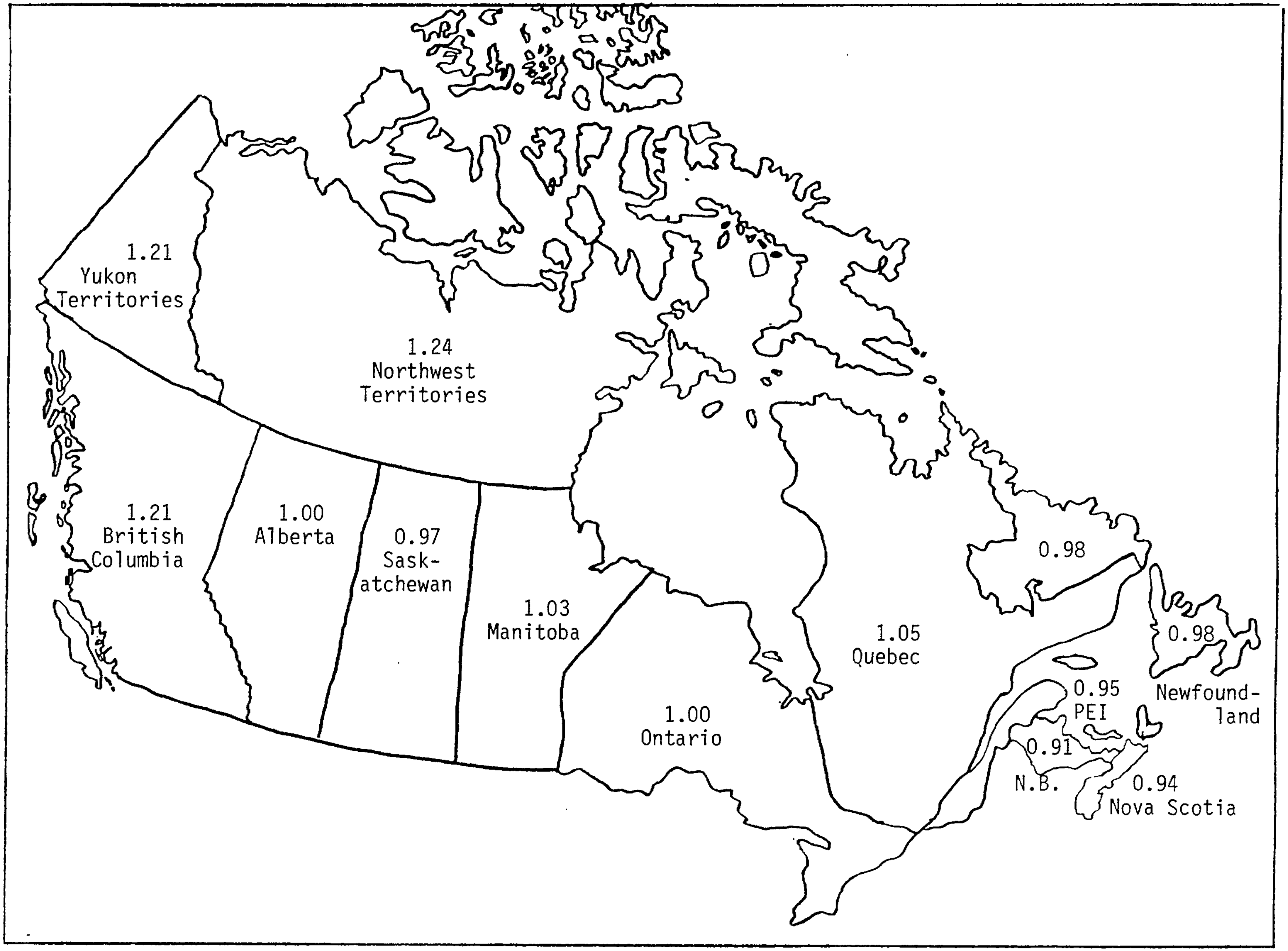


FIGURE 4 - 2

OPERATING COSTS

BLANKET REGIONAL COST FACTORS

4.3 COMPONENT COST FACTORS

	Cost Components					
Area	Labour	Plant & Equipment	Mat'ls. & Consumables	Hydro Power	Transportation	Provincial Tax
British Columbia	1.34	1.00	1.00	0.92	1.75	1.20
Alberta	1.05	1.00	0.92	0.97	1.50	-
Saskatchewan	0.97	1.00	0.93	1.09	1.50	1.29
Manitoba	1.04	1.00	0.95	0.66	2.00	1.29
Ontario	1.00	1.00	1.00	1.00	1.00	1.00
Quebec	1.06	1.00	1.04	0.94	1.00	1.41
Newfoundland	0.86	1.00	1.05	1.29	1.75	1.89
New Brunswick	0.82	1.00	0.95	1.17	1.35	2.36
Nova Scotia	0.85	1.00	0.98	1.21	1.70	1.37
Prince Edward Island	0.82	1.00	1.02	1.21	2.00	1.71
Yukon Territory	1.14	1.00	1.15	1.51	3.75	-
Northwest Territories	1.19	1.00	1.17	1.51	3.50	-

4.4 WORKED EXAMPLES

Capital Cost Regional Cost Factor

Cost Classification Area	Regional Cost Factor for Manitoba (from Table 4.3)		Percentage Distribution of Capital Costs (determined by User)		Multiplication Product	
Labour	1.04	x	40	% =	0.42	(a)
Plant & Equipment	1.00	x	20	% =	0.20	(b)
Materials & Consumables	0.95	x	35	% =	0.33	(c)
Hydro Power	0.66	x	0	% =	0.00	(d)
Transportation	2.00	x	3	% =	0.06	(e)
Provincial Tax	1.29	x	2	% =	0.03	(f)
			Total = 100%			

Capital Cost Regional Cost Factor = sum (a to f) = 1.04

Operating Cost Regional Cost Factor

Cost Classification Area	Regional Cost Factor for Manitoba (from Table 4.3)		Percentage Distribution of Operating Costs		Multiplication Product	
Labour	1.04	x	54	% =	0.56	(g)
Plant & Equipment	1.00	x	0	% =	0.00	(h)
Materials & Consumables	0.95	x	42	% =	0.40	(i)
Hydro Power	0.66	x	0	% =	0.00	(j)
Transportation	2.00	x	3	% =	0.06	(k)
Provincial Tax	1.29	x	1	% =	0.01	(l)
			Total = 100%			

Operating Cost Regional Cost Factor = sum (g to l) = 1.03

SECTION 5

5.0 MINERAL DEPOSIT VALUE

5.1 INTRODUCTION

The purpose of this section is to assist the user who has not already developed preliminary estimates of tonnage, grade and economic value from the available geological information. These calculations are to be completed using Forms 5(a) to 5(f).

The basic approach is outlined below:

i) Calculate - geological tonnes and grade
  - mineable tonnes and grade

ii) Estimate mill recovery

iii) Estimate the values recovered by the mill

iv) Estimate net value after smelting.

5.2 TONNAGE/GRADE CALCULATIONS

The reserve calculation method selected for any particular deposit will be dependent upon the geological and engineering elements unique to each deposit, and no one technique is universally applicable.

Reserve calculation methods available include: calculation by mining block, calculation by polygons, calculation by triangles, geostatistical techniques, and the classical, calculation by section, which is discussed in this Section.

5.2.1 Geological Tonnes and Grade

i) Assemble the most up-to-date geological data, diamond drill intersections (length and assay values) and other relevant information.

ii) Prepare the following drawings showing all of the available geological information:

  a) surface plan
  b) cross-sections
  c) longitudinal section(s)

iii) On each cross-section indicate the geological limits of mineralized areas. A preliminary cut-off grade may have to be arbitrarily selected. Limit the influence of each intersection to 50% of the distance to the next intersection.

Each cross-section therefore has one or more mineralized areas shown, each with an identifiable area and grade.

Assessing the average width or thickness is extremely important as it will influence the selection of a mining method in Section 2.0.

iv) From the longitudinal section, determine the strike length that each cross-section will represent.

v) For each cross-section, multiply each area of mineralization by the strike length identified in iv) above to determine a volume in cubic metres.

vi) Multiply the volume of each area by a tonnage factor (see Note 1) to determine the tonnage of mineralization.

vii) Multiply the tonnage of each area by its grade, expressed in units of metal per tonne, (kg of base metal or grams of precious metal), to determine total units of metal.

viii) For each cross-section, calculate the total tonnes and total units of metal represented by that cross-section by adding each mineralized area.

ix) Calculate the average grade of each cross-section by dividing total units of metal by total tonnes.

x) Add the total tonnes from each cross-section to determine the total geological reserve.

xi) Add the total units of metal from each cross-section and divide by the total geological reserve to determine the overall average grade.

Note 1 "Tonnage Factor" (tonnes/m^3)

The tonnage factor is a factor used to convert volume to tonnage. Use the actual tonnage factor if known. If the tonnage factor is not known refer to the table below.

Type of Mineralization	Tonnage Factor
Gold	2.7 tonnes/m^3
Disseminated Base Metal	3.2 tonnes/m^3
Massive Sulphides	4.0 tonnes/m^3
Waste	2.7 tonnes/m^3

5.2.2 Mineable Tonnes and Grade

In general, mineable tonnes are developed by superimposing a mine design over the outline of the geological reserves. This involves selecting a suitable mining method(s) which is part of Section 2.0 "Operating Costs".

The selection made will influence the ability to mine selectively and may require some waste to be mined or, conversely, some mineralization to be left behind.

In this manual, mineable tonnes and grade are developed in three distinct steps as follows:

a) In-situ reserves are determined for each geological cross-section on Form 5(d).

b) Total in-situ reserves for the mine are then tabulated on Form 5(c).

c) Finally, total mineable tonnes and grade are calculated on Form 5(e) by manipulating the numbers generated on Form 5(c), to adjust for pillars, dilution and stope losses.

The in-situ reserve tonnes and grade can be calculated in a similar way to geological tonnes and grade.

The user must consider the implications of the selected mining method and place mining limits on the geological sections developed in Section 5.2.1. At this stage, the geological grade may be diluted because some waste has to be mined. There is no regard for pillars at this point.

Thereafter, the true mineable reserves are determined by adjusting the in-situ reserves for three additional considerations:

- mining recovery factor (Note 2)
- dilution factor (Note 3)
- stope losses factor (Note 4)

The "mining recovery factor" accounts for all of the solid ore which must be left in place and never mined. This may include crown pillars, sill pillars, structural support pillars and other mineralization that is abandoned for geotechnical or economic reasons. Refer to Note 2 on the following page.

The ore actually recovered will be diluted by waste rock and/or adjacent fill material. The amount of dilution will depend on the mining method and the ground conditions. Refer to Note 3 on the following page.

Finally, some of the tonnage broken may not be recovered. The amount not recovered will depend mainly on the mining method selected. Refer to Note 4 on the following page.

Note 2 "Mining Recovery Factor"

The user is cautioned that this factor is very difficult to evaluate accurately until a detailed mine design is completed.

Mining Method	Mining Recovery Factor	
	Range	Typical
Blasthole	60% - 100%	80%
Cut & Fill	70% - 100%	85%
Shrinkage	75% - 100%	90%
Room & Pillar	50% - 75%	60%

Note 3 "Mining Dilution Factor"

The mining dilution factor expresses the diluted tonnes as a factor of the in-situ reserve tonnes.

MINING METHOD	Expected Ground Conditions		
	Excellent	Average	Poor
Blasthole	1.20	1.30	N/A
Cut and Fill	1.05	1.10	1.15
Shrinkage	1.10	1.15	1.25
Room and Pillar	1.05	1.10	1.20

Note 4 "Stope Losses Factor"

This factor expresses tonnes actually recovered as a factor of diluted tonnes.

Mining Method	Stope Losses Factor
Blasthole	0.8 to 1.0
Cut and Fill	1.0
Shrinkage	0.9 to 1.0
Room and Pillar	1.0

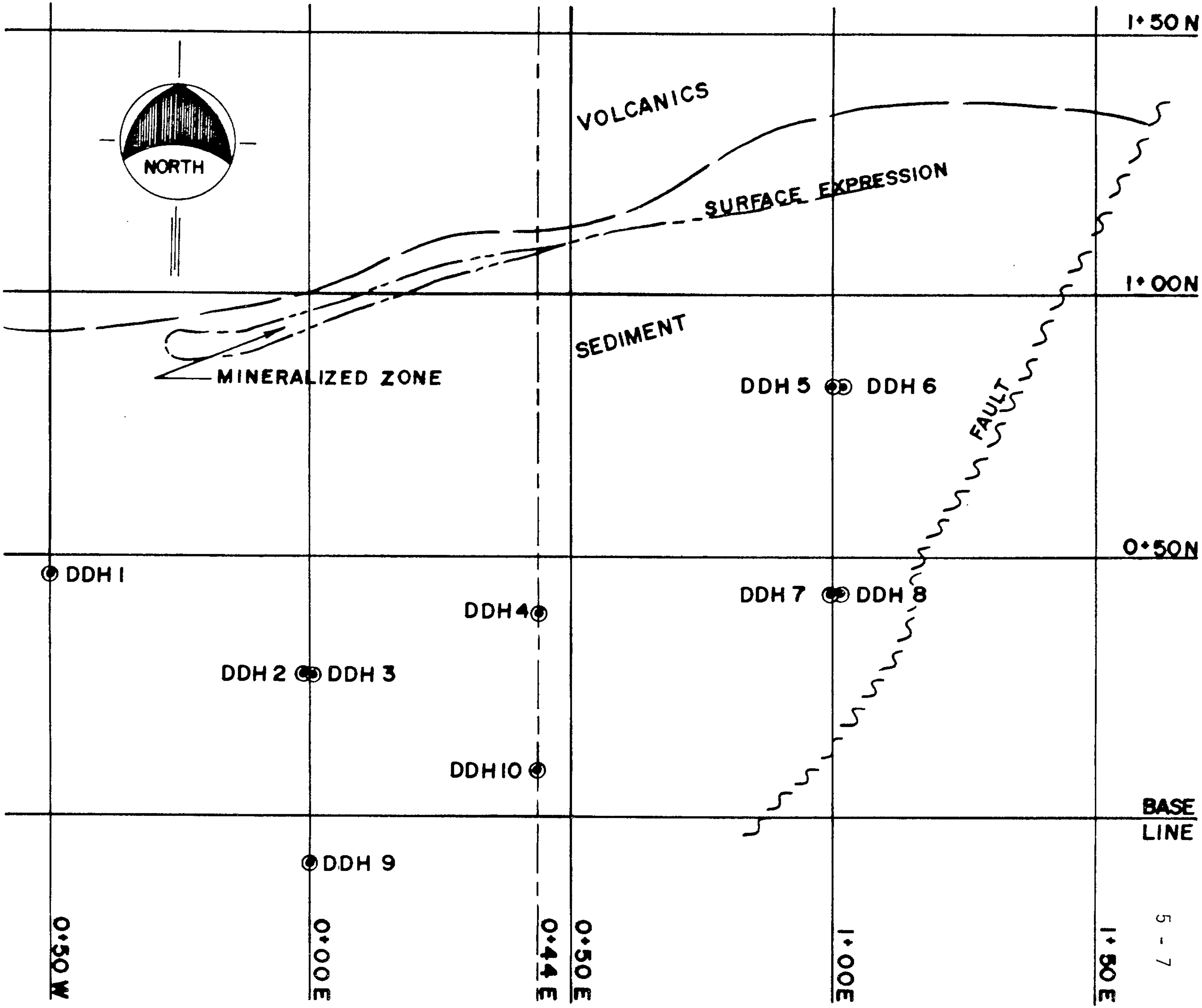
5.2.3 Worked Example

The following drawings and calculations have been developed for a gold/silver deposit, using shrinkage stoping in average ground conditions.

For simplicity, calculations are completed for one cross-section only.

It should be noted that, in this example, all of the geological reserve tonnage falls within the mining limits. This will not always be the case and some mineralization may be left behind.

GEOLOGICAL SURFACE PLAN



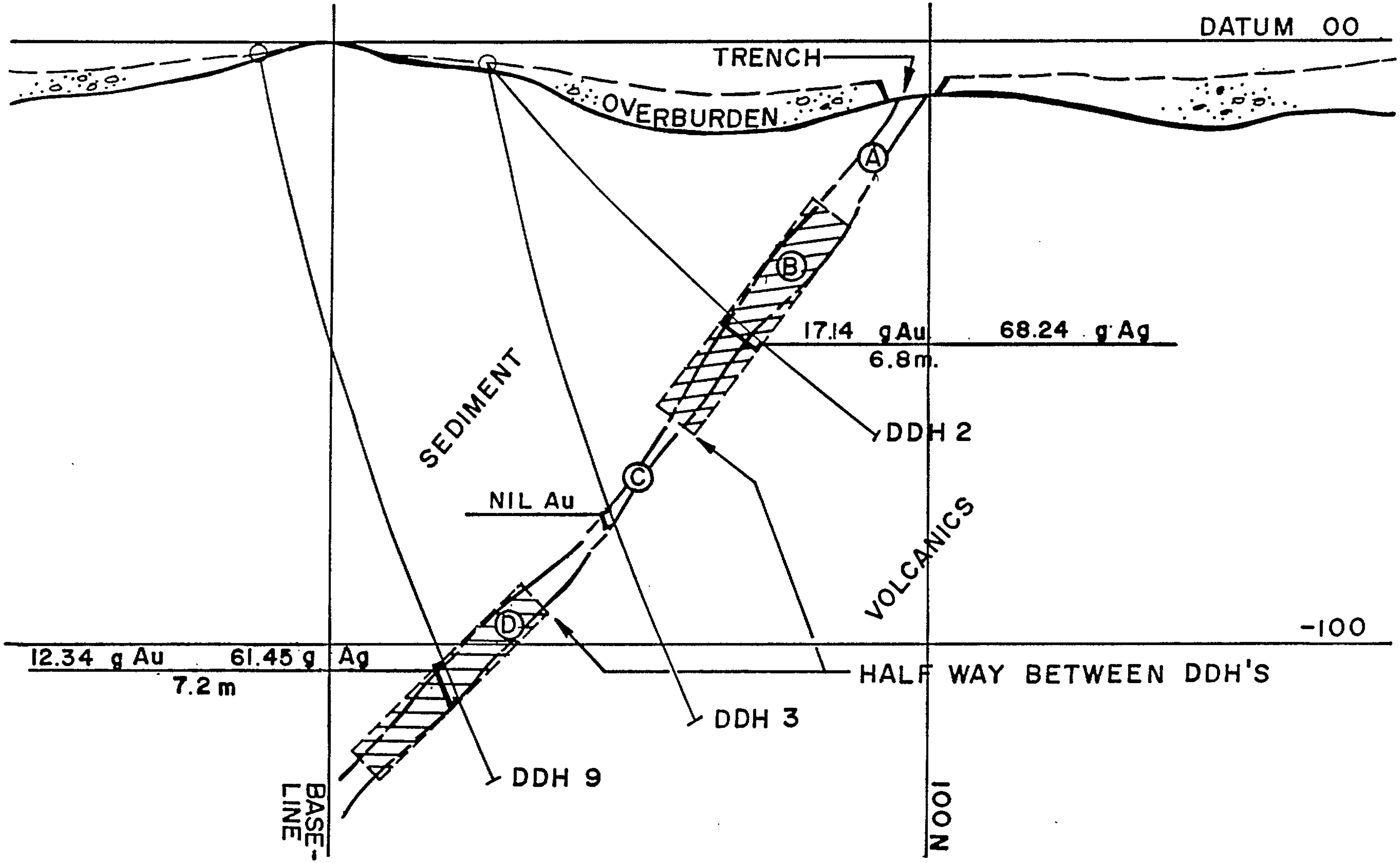
DWG. NO.

SCALE 1:1000

DATE APRIL 1986

CROSS SECTION @ 0+00E

-LOOKING WEST-



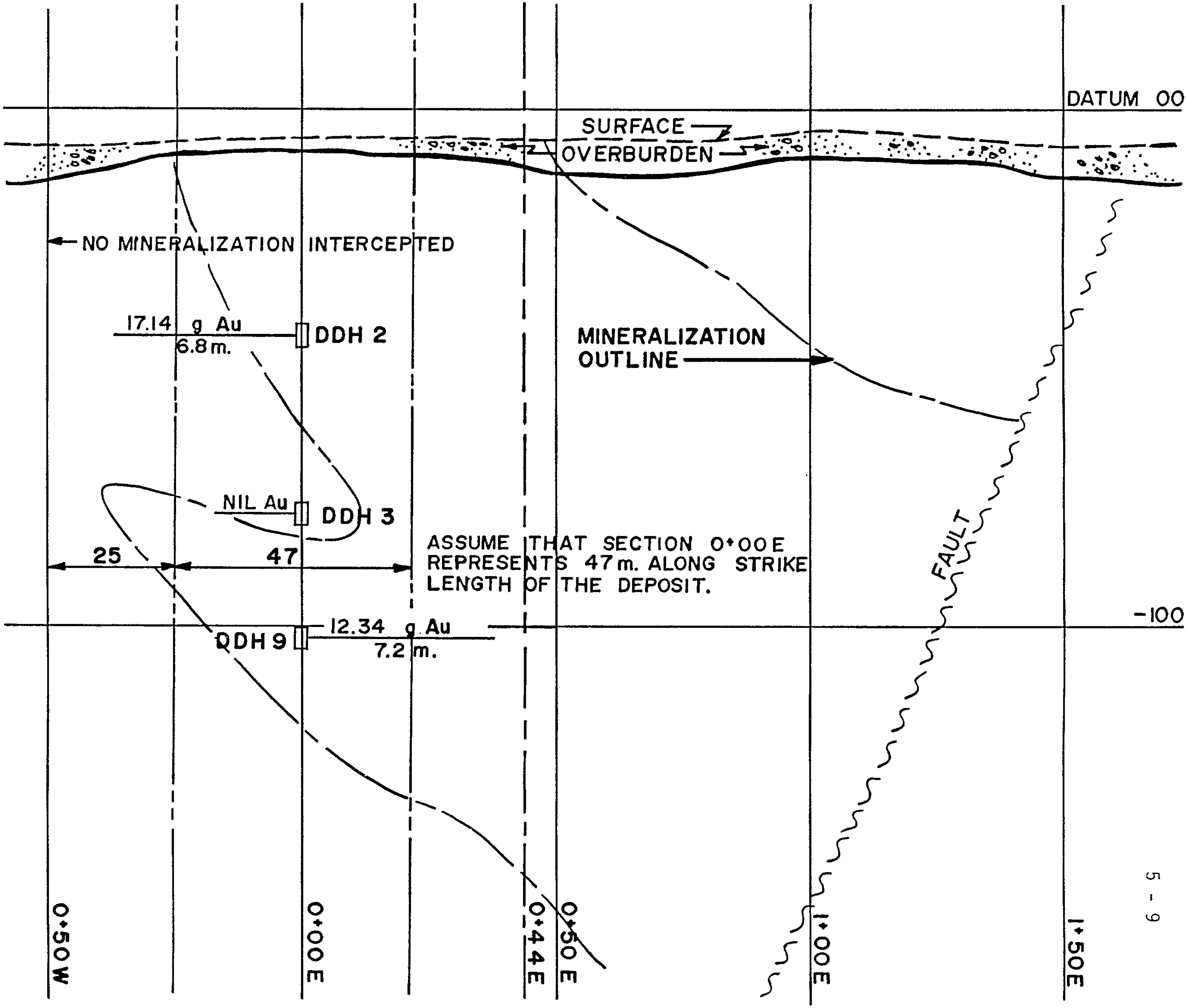
DWG.NO.

SCALE 1:1000

DATE APRIL 1986

LONGITUDINAL PROJECTION

-LOOKING NORTH-



DWG. NO.

SCALE 1:1000

DATE APRIL 1986

FORM 5(a) EXAMPLE

GEOLOGICAL TONNES & GRADE

SUMMARY (see next page for calculation)		Unit of Measurement
Primary metal is	Gold (Au)	g/tonne
Secondary metal is	Silver (Ag)	g/tonne
Tertiary metal is	N/A	N/A

Cross Section No.	Total Tonnes of Mineralization	Total Units of Metal		
		Primary Metal	Secondary Metal	Tertiary Metal
0 + 00E	41,242	627,702	2,702,340	N/A
	(A)	(B)	(C)	(D)
TOTALS	41,242	627,702	2,702,340	N/A

TOTAL GEOLOGICAL RESERVE = 41,242 (A)

GEOLOGICAL RESERVE GRADE:

Primary Metal (B/A) = 15.22 g/tonne

Secondary Metal (C/A) = 65.52 g/tonne

Tertiary Metal (D/A) = N/A

FORM 5(b) EXAMPLE

GEOLOGICAL TONNES & GRADE

MINERAL RESERVE BY CROSS-SECTION

Cross-Section No. 0 + 00E

	Mineralized Area			
	A	B	C	D
Diamond Drill Hole No.		2		9
Area of Mineralization (m^2)		195		130
Grade of Mineralized Area:				
Primary Metal		17.14		12.34
Secondary Metal		68.24		61.45
Tertiary Metal		-		-
Strike Length Represented by Cross-section (m)		47		47
Volume of Mineralization (m^3)		9,165		6,110
Tonnage Factor (tonnes/m^3)		2.7		2.7
Tonnes of Mineralization		24,745		16,497
Total Units of Metal:				
Primary Metal		424,129		203,573
Secondary Metal		1,688,599		1,013,741
Tertiary Metal		-		-

TOTAL TONNES (A + B + C + D) = 41,242

	Total Units of Metal	Average Grade
Primary Metal	627,702 g	15.22 g/tonne
Secondary Metal	2,702,340 g	65.52 g/tonne
Tertiary Metal	-	-

FORM 5(c) EXAMPLE

IN-SITU TONNES & GRADE

SUMMARY - From Forms 5(d)

		Unit of Measurement
Primary metal is	Gold (Au)	g/tonne
Secondary metal is	Silver (Ag)	g/tonne
Tertiary metal is	N/A	N/A

Cross Section No.	Total In-Situ Tonnes	Total Units of Metal		
		Primary Metal	Secondary Metal	Tertiary Metal
0 + 00E	47,334	627,702	2,702,340	N/A
	(A)	(B)	(C)	(D)
TOTALS	47,334	627,702	2,702,340	N/A

TOTAL IN-SITU RESERVE = 47,334 (A)

IN-SITU RESERVE GRADE:

Primary Metal (B/A) = 13.26 g/tonne

Secondary Metal (C/A) = 57.09 g/tonne

Tertiary Metal (D/A) = N/A

FORM 5(d) EXAMPLE

IN-SITU TONNES & GRADE

IN-SITU RESERVE BY CROSS-SECTION

Cross-Section No. 0 + 00E

	Mining Zone			
	A	B	C	D
AREAS (Within Mining Limits)				
Area of Mineralization (m^2)		195		130
Area of Waste (m^2)		33		15
Total Area (m^2)		228		145
VOLUMES (Within Mining Limits)				
Strike Length Represented by Cross-section (m)		47		47
Volume of Mineralization (m^3)		9,165		6,110
Volume of Waste (m^3)		1,551		705
Total Volume (m^3)		10,716		6,815
TONNES (Within Mining Limits)				
Tonnage Factor - Ore (tonnes/m^3)		2.7		2.7
Tonnage Factor - Waste (tonnes/m^3)		2.7		2.7
Tonnes of Mineralization		24,745		16,497
Tonnes of Waste		4,188		1,904
Tonnes of Mineralization & Waste (Ore)		28,933		18,401
TOTAL TONNES OF ORE - ALL ZONES		47,334	(a)	
GRADES OF MINERALIZED AREAS (Geological Grade)				
Primary Metal		17.14		12.34
Secondary Metal		68.24		61.45
Tertiary Metal		-		-
UNITS OF METAL (Tonnes of Mineralization x Geological Grade)				
Primary Metal		424,129		203,573
Secondary Metal		1,688,599		1,013,741
Tertiary Metal		-		-

	Total Units All Zones (b)	Average Grade All Zones (within mining limits) (b ÷ a)
Primary	627,702	13.26 g/tonne
Secondary	2,702,340	57.09 g/tonne
Tertiary	-	-

FORM 5(e) EXAMPLE

MINEABLE TONNES AND GRADE TO MILL

ADJUSTMENT FOR MINING RECOVERY

			Total In-Situ Reserve		Mining Recovery Factor		Actually Mined	
Tonnes		=	47,334	x	0.90	=	42,600	(a)
Primary Metal	- Units	=	627,702	x	0.90	=	564,932	(b)
	- Grade	=	13.26				13.26	(c)
Secondary Metal	- Units	=	2,702,340	x	0.90	=	2,432,106	(d)
	- Grade	=	57.09				57.09	(e)
Tertiary Metal	- Units	=	N/A	x	0.90	=	N/A	(f)
	- Grade	=	N/A				N/A	(g)

ADJUSTMENT FOR DILUTION

Dilution Factor = 1.15 (h)

Diluted Tonnes (a x h) = 48,990 (i)

"MINEABLE GRADES"

Primary Metal (c/h) = 11.53 (j)

Secondary Metal (e/h) = 49.64 (k)

Tertiary Metal (g/h) = N/A (l)

ADJUSTMENT FOR STOPE LOSSES

Stope Losses Factor = 0.90 (m)

"MINEABLE TONNES" to Mill (i x m) = 44,091 (n)

Units of Metal/Tonne in Mill Feed:

Primary Metal (b/i) = 11.53

Secondary Metal (d/i) = 49.64

Tertiary Metal (f/i) = N/A

5.3 ESTIMATE OF VALUE

5.3.1 Mill Recovery

Section 5.2 calculated the tonnage and grade actually mined and delivered to the mill.

A mill facility will not extract 100% of the units mined, therefore, a mill recovery factor is required to adjust the units recovered by the mining operation.

If metallurgical test work concerning mill recovery is available then use that information. Should no suitable information be available, select a factor from the table below:

Note that the recovery factor for the primary metal is likely to be higher than for the secondary metal.

Type of Mineralization	Mill Recovery Factor
Precious Metal	0.80 to 0.97
Base Metal	0.75 to 0.95
Complex Base Metal	0.60 to 0.85
Other	0.50 to 0.80

Multiply the units of metal in the mill feed by the mill recovery factor to determine the units of metal per tonne recovered by the mill.

5.3.2 Values Recovered by Mill

The value of a tonne of ore after milling can be established by multiplying the mineable grade by the mill recovery factor by the current price (or projected price if available) per unit.

Current metal prices can be obtained from the Northern Miner; the Engineering and Mining Journal; London Metal Exchange; Hardy and Harmon (New York).

5.3.3 Net Value after Smelting or Refining

The revenue actually realized by the mine operator will depend on the contract negotiated with a smelter or the specific charges related to refining a particular bullion product.

Each contract is unique and returns cannot be generalized. It is essential to realize, however, that the revenue received by the mine operator will only be a percentage of the total value of the metal contained in the concentrate.

If inquiries have been made with regard to smelter returns or refining charges, adjust the values contained in the concentrate accordingly to arrive at:

NET VALUE AFTER SMELTING/REFINING

5.3.4 Worked Example

The following example carries on from the example of tonnage and grade calculations in Section 5.2.

The calculation develops a net value per tonne of mineable ore delivered to the mill.

The "Net Value After Smelting" allocated in the example is for illustrative purposes only and is not a calculated figure.

FORM 5(f)

EXAMPLE

MINERAL DEPOSIT VALUE

	Primary Metal	Secondary Metal	Tertiary Metal
Units of Metal/Tonne in Mill Feed	11.53	49.64	-
Mill Recovery Factor	0.90	0.90	-
Units of Metal Recovered Per Tonne of Mill Feed	10.38	44.68	-
Current Metal Price/Unit	$15.11/g ($470 Cdn/oz)	$0.23/g ($7.15 Cdn/oz)	-
Value Per Tonne of Ore after Milling	$156.84	$10.28	-

TOTAL VALUE PER TONNE OF ORE AFTER MILLING = $167.12

NET VALUE PER TONNE OF ORE AFTER SMELTING/REFINING (Approximation only) = $155.00

5.4 POTENTIAL PROBLEMS IN ESTIMATING MINERAL DEPOSIT VALUE

Tremendous errors can be made in estimating a mineral reserve. There have been instances where mineral reserve estimates have been significantly inaccurate even after detailed drilling and analysis. This section is provided to alert the user to some of the areas where major problems could arise.

Sources of Error

Area	Potential Problem
Conversion of Units	Imperial & Metric units not converted correctly.
Geology	Geology misinterpreted.
	Mineral values are erratically distributed throughout the deposit so that a mineral reserve estimate may be incorrect.
	The deposit could be cut-off by structural features such as a fault.
	The deposit may be complex both physically and mineralogically.
Geotechnical	May require more or larger pillars than anticipated.
	Dilution may be significantly higher.

Sampling	Assay values may be misleading. Sampling procedures may be deficient.
Optimism	Assumptions may be too optimistic.
Actual Mining Conditions	Actual conditions experienced underground may be significantly different than expected and the tonnage that can be practically mined may be less than expected.
Metallurgical Problems	For various reasons, concentrator recovery may not be as high as expected.
Economic Mining Grade (Cut-Off Grade)	The preliminary mineral inventory estimate is not based on any minimum acceptable economic grade of mineralization. In a detailed feasibility study a minimum acceptable grade of mineralization would be calculated and used to decide what areas are to be considered mineable.
Metal (Product) Prices	Changes in metal (product) prices may have a significant effect on the economic viability of the deposit.
Smelter Contract	Change in market.

SECTION 6

6.0 PRELIMINARY CASH FLOW SUMMARY

6.0 PRELIMINARY CASH FLOW SUMMARY

General

The user is now in a position to make a very preliminary assessment of the economic viability of his deposit.

He has chosen a mining method and mining rate, developed operating and capital costs and calculated a Net Value After Smelting/Refining. A simple cash flow chart can now be developed showing the cash out-flows and in-flows for each year of operation.

Costs generated or selected in the preceding sections of this manual are brought together on an "Economic Analysis" form or flow chart to provide the total profitability picture. With a little effort on the part of the user, particularly in breaking down certain cost groupings into fixed and variable, this analysis form can become a handy tool for executing certain sensitivity evaluations.

6.1 CASH FLOW CHART

A simple cash flow chart developed for the first five years of operation, for a small mine will generally show whether the project has the potential to be viable or not.

Form 6 presents a format for determining cash flow by year for the first five years of operation.

The annual cash flows calculated do not consider depreciation and taxes. The information developed can be used to perform a Discounted Cash Flow Rate of Return (D.C.F.R.O.R.) calculation.

6.2 OTHER APPROACHES

If the deposit does not generate a profit, return to Section 2.0 and reconsider some of the key decisions.

Other possible approaches may include:

1) Mining only the higher grade sections of the deposit.

2) Using another means of mine access.

3) Selecting a different mine production rate.

4) Using a different mining method.

5) Reducing the initial capital outlay.

6) Custom milling.

7) Undertaking additional work to increase tonnage and/or grade.

8) Hiring a contractor to do the production mining and provide all of the capital equipment.

A deposit that still does not show a profit, even after looking at other approaches, may not necessarily be uneconomic.

Regardless of what your conclusions are at this point, read the next section as it deals with evaluating results.

6.3 EVALUATION OF RESULTS

6.3.1 Discussion

Many assumptions have been made by the user while working his way through this manual which have a significant effect on the conclusion reached.

The selection and/or calculation of key parameters such as mineable tonnes and grade, production rate, mining method, price of product, as well as capital and operating costs are all critical to the overall economics of the project. Form 6 will help the user identify the order of magnitude cost improvement that is required to achieve break-even or better. By selectively applying the plus or minus 30% accuracy range projected for this manual to specific cost categories, the user should be able to compute a worst and best case scenario from Form 6. Actual sensitivity analysis will require recalculating the primary forms used to generate Form 6, based on changed parameters.

A mineral deposit which appears to be marginal may prove to be viable with a change in the variables but only a proper detailed feasibility study will provide definitive answers.

6.3.2 Level of Confidence in Information and Assumptions.

Regardless of the results obtained, good or bad, the user should evaluate his own level of confidence in each step by the estimating process. By completing the following assessment honestly, an overall confidence in the results can be established.

Area	High	Moderate	Low
Understanding of the evaluation process used in this manual	_____	_____	_____
Understanding of the geology of the deposit and the mineral reserve estimate.	_____	_____	_____
Understanding of required mineral processing.	_____	_____	_____
Level of confidence in your Capital Cost Estimates	_____	_____	_____
Level of confidence in your Operating Cost Estimates	_____	_____	_____
Level of confidence in the metal price used.	_____	_____	_____
Level of confidence in the marketing and saleability of the product.	_____	_____	_____
Overall confidence based on the preceding.	_____	_____	_____

Review responses above and subjectively relate them to the results obtained on Form 6.

The use of this manual and its estimating process provides a very preliminary assessment of the profitability of the mineral deposit examined. It should be recognized that, at best, this examination can only result in a rough approximation of the economic model involved. It can however, serve to point out the direction to be followed in pursuing the ultimate evaluation and exploitation of the resource.

SECTION 7

7.0 EXPLORATION PROGRAMMES

General

This section explains how the manual can be used to cost the construction and excavation required to conduct an underground exploration programme.

It is important to understand that capital costs presented in Section 3.0 are "all - inclusive" and do not require additional support or services to make them complete.

For example, the cost per metre for shaft sinking includes the cost of a temporary camp, electrical power, compressed air and all other support services required.

Therefore, to cost an exploration programme the user has only to identify the work required and select the appropriate costs from Section 3.0 accordingly. Form 3(a) can be used to develop the costs.

Each section of capital costs is briefly described on the following pages and comments are made as to the relevance of each section to an exploration programme. In some sections, only a portion of the costs indicated in Section 3.0 are applicable. In these cases, the costs indicated in Section 3.0 should be multiplied by the percentages listed in this section.

7.1 CAPITAL COSTS FOR EXPLORATION PROGRAMMES

CAPITAL COSTS SECTION NO.	TITLE AND DESCRIPTION	RELEVANCE TO EXPLORATION PROGRAMME
3.1	Introduction and Criteria	All criteria apply.
3.2	Feasibility Studies and Detailed Engineering	Only a portion of this work is required for exploration. Use 50% of indicated cost
3.3	Additional Diamond Drilling and Sampling	Fully applicable to exploration programme.
3.4	Permits and Environmental Studies	Only permitting applicable to exploration programme. Use 10% of indicated cost
3.5	Project Management and Preproduction Scheduling Cost of Owner's site management	Fully applicable to exploration programme.
3.6	Access to Minesite Cost of providing new or upgraded road and bridges	Applicable but quality of road required may be reduced. Adjust cost/km accordingly.
3.7	Site Preparation Cost of clearing, filling and grading site	Cost for a given area still applicable. The area required for exploration will be less than for production. Use 50% of indicated cost

CAPITAL COSTS SECTION NO.	TITLE AND DESCRIPTION	RELEVANCE TO EXPLORATION PROGRAMME
3.8	Camp Installation	Not applicable.
	Camp for production phase of project	Construction and excavation costs include temporary camp costs.
3.9	Site Services	Applicable but quality and size of facilities may be reduced.
	Cost of providing water supply, sewage handling, storage, communications, etc.	Use 70% of indicated cost
3.10	Electrical Power & Compressed Air	
	Electrical Power - cost of constructing powerline or installing generators. (For the production phase.)	If estimates of capital costs are calculated using hydro power and a line is not already built to the site then include the cost of a powerline. If generated power is used, the applicable capital costs include generator rental and operating costs where applicable.
	Compressed air - plant for production phase.	Not applicable. Construction and excavation costs include temporary compressor costs.

CAPITAL COSTS SECTION NO.	TITLE AND DESCRIPTION	RELEVANCE TO EXPLORATION PROGRAMME
3.11	Offices, Shops, Dry, Warehouse	Not applicable. The Owners representative(s) may use part of the Contractors office. If the user considers a separate office necessary include an allowance of $15,000.00
3.12	Mine Access Cost of shaft, decline or adit including mobilization, setup, collar or portal, etc.	Fully applicable. Costs include all support services.
3.13	Ancillary Shaft Excavations & Installations Station & pocket excavation & construction	Fully applicable. Could delay loading pocket installation but usually cheaper if installed at the time the shaft is excavated.

CAPITAL COSTS SECTION NO.	TITLE AND DESCRIPTION	RELEVANCE TO EXPLORATION PROGRAMME
3.14	Hoisting Systems, Headframes & Bins	
	Hoist & hoisthouse	Fully applicable. Alternately could rent from Contractor with an option to purchase. In this case, allow 50% of indicated capital cost and a monthly rental of 5% of the difference or 2.5% of the total indicated cost.
	Headframe & collarhouse	Fully applicable. Exception to above is if temporary mobile headframe used (max. height 15 m). In this case, allow $20,000 for temporary collarhouse and a headframe rental of $2,500 per month.
	Bin or dump area	Allow for dump area only.
	Conveyances	Fully applicable. Car hoisting will not affect costs appreciably.

CAPITAL COSTS SECTION NO.	TITLE AND DESCRIPTION	RELEVANCE TO EXPLORATION PROGRAMME
3.15	Ventilation & Mine Air Heating	Not applicable.
	Primary ventilation & heating	Sufficient ventilation & heating for exploration will be supplied by Contractor.
3.16	Underground Development	Only "Level Development" (3.16.1) applicable to exploration. Establish quantity and multiply by unit costs. See Appendix 3.B for unit rates.
3.17	Underground Installations	Allow for main sumps and pumps only.
	Main sumps and pumps, rockbreaker and grizzly, ore pass controls, electrical substations & misc. installations.	
3.18	Equipment	Not applicable.
	Mining and support equipment	Unit prices for excavation and construction include contractor's equipment rentals.

CAPITAL COSTS SECTION NO.	TITLE AND DESCRIPTION	RELEVANCE TO EXPLORATION PROGRAMME
3.19	<u>Concentrator</u>	Not applicable.
3.20	<u>Cost Contingency</u>	Fully applicable.
3.21 to 3.24	<u>Ongoing Capital Costs</u>	Not applicable.

APPENDIX A

EXAMPLE ONE

FORM 1

BASIC INFORMATION

Estimate Prepared By: John Smith Date: June 13, 1986

Name of Property: PRIME EXAMPLE GOLD MINE

Property Location: Northern British Columbia, 100 miles west of Dawson Creek

Brief Description of Site and Local Area: Uneven rolling ground with rocky outcrops

Expected Overburden Conditions (Depth and Type): 3 metres of dry unconsolidated glacial till

Expected Rock Conditions: Competent ore with reasonable to good hanging wall and footwall

Expected Ground Water Conditions: Dry

Other Relevant Information: Ore horizon - 100 m to 300 m below surface, strike length 300 m; ore thickness 3 m; dip 80°

Geological tonnes and grade:

481,342 tonnes - 11.45 grams/tonne Au (see Form 5(a))

- 50.50 grams/tonne Ag

Attach additional sheets as required.

FORM 2(a)

OPERATING COSTS

SUMMARY (costs are developed on the next two pages)

		$/Tonne	
Stoping Costs		21.29	
Hoisting or Ramp Haulage Cost		2.80	
Level Haulage Cost		2.40	
General Mine Expense		4.40	
Surface Plant and Mine Services		16.88	
Staff and Administration		8.00	
Milling		17.75	
	Subtotal	73.52	
Add Cost Contingency @ 13%		9.56	
	Subtotal	83.08	(a)
Regional Operating Cost Factor	(blanket factor)	1.21	(b)
TOTAL OPERATING COST (a) x (b)		100.53	
Transportation of Mine Product		N/A	

FORM 2(a)

OPERATING COSTS

DETAILED CALCULATION FORM

Reference Section No.	Item	Operating Cost $/Tonne Ore
2.2	Selection of Production Rate	
	Rate selected 260 tonnes/day	
	Mining days per year 350	
	Mining shifts per day 2	
2.3	Selection of Mining Method	
	Method selected Shrinkage	
2.4	Stoping Costs	21.29
2.5	Selection of Mine Access and Haulage Method	
	Access selected Shaft	
	Level haulage selected Track	
	Depth 300 m	
2.6	Hoisting or Ramp Haulage Cost	2.80
2.7	Level Haulage Cost	
	Haulage Distance 150 m	
	Haulage Capacity/Trip 20 tonnes	
	Cost	2.40
2.8	General Mine Expense	4.40

2.9 Surface Plant and Mine Services

Power source selected Hydro

Cost

a)	Labour	4.40	
b)	Materials and Operating Costs	1.70	
c)	Power	5.00	
d)	Camp	5.75	
e)	Road Maintenance	0.03	
	Total Surface Plant and Mine Services		16.88

2.10 Staff and Administration 8.00

2.11 Milling

Selection of Location On-Site

Cost 17.75

2.14 Transportation of Mine Product

i) Bullion N/A

ii) Ore (enter cost from graph) N/A

iii) Concentrate ______ $/tonne (a)
(enter cost from graph)

Concentrating Ratio ______ (b)

Cost/tonne mined (a) ÷ (b) N/A

3.20 Cost Contingency

3.20.1	Contingency for items omitted	5 %	
3.20.2	Contingency for variations in conditions	5 %	
3.20.3	Contingency for delays due to location	3 %	
	Total Contingency Percentage, (add 3 lines above)		13 %

Transfer all subsection totals to the summary page.

FORM 2(b)

MANPOWER SCHEDULE

Reference Section No.	Item			Manpower
3.21	Ongoing Capital Development			2
2.4	Stoping			
	Mining method selected	Shrinkage		
	Productivity (tonnes/manshift)	21	(a)	
	Production rate (tonnes/day)	260	(b)	
	Manpower required (b)/(a)			13
2.6	Hoisting or Ramp Haulage			
	Hoisting:			
	Shifts worked per day	2	(c)	
	Manpower required per shift	2	(d)	
	Manpower required per day (c) x (d)			4
	Ramp:			
	Vertical depth		(e)	
	Manpower required			N/A
2.7	Level Haulage			
	Haulage method selected	Track/20T		
	Manpower required			4
2.8	General Mine Expense			
	Track or trackless mine	Track		6
	Subtotal Underground Manpower (including hoistman)			29
2.9	Surface Plant and Mine Services			7
2.10	Staff and Administration			9
2.11	Milling			17
	Subtotal Surface Manpower			33
	TOTAL ON-SITE MANPOWER			62

FORM 3(a)

PREPRODUCTION CAPITAL COSTS

SUMMARY (costs are developed on the next 8 pages)	$	
Feasibility Studies and Detailed Engineering	255,000	
Additional Diamond Drilling and Sampling	79,500	
Permits and Environmental Studies	175,000	
Project Management and Preproduction Scheduling	152,000	
Access to Minesite	625,000	
Site Preparation	180,000	
Camp Installation	395,000	
Site Services	62,000	
Electrical Power & Compressed Air	510,000	
Offices, Shops, Dry, Warehouse	189,000	
Mine Access	1,869,000	
Ancillary Shaft Excavations & Installations	549,500	
Hoisting Systems, Headframes & Bins	976,000	
Ventilation & Mine Air Heating	48,000	
Underground Development	1,838,600	
Underground Installations	498,500	
Equipment	590,000	
Concentrator	3,780,000	
Subtotal	12,772,100	
Add Cost Contingency @ 13 %	1,660,373	
Subtotal	14,432,473	(a)
Regional Capital Cost Factor (blanket factor)	1.16	(b)
TOTAL PREPRODUCTION CAPITAL COST (a) x (b)	16,741,669	
USE	16,742,000	

PREPRODUCTION CAPITAL COSTS - Detailed Calculation Form

Reference Section No.	Item		Capital Cost $
3.2	Feasibility Studies & Detailed Engineering		
	For shaft or ramp access?	Shaft	
	Subsection Total (enter cost from graph)		255,000
3.3	Additional Diamond Drilling & Sampling		
3.3.1	Drilling from surface:		
	Number of holes	None	
	Average hole length	m	
	Cost/metre	$ /m	
	Subtotal a)	$	
3.3.2	Underground drilling:		
	Number of holes	30	
	Average hole length	50 m	
	Cost/metre	$ 45.00 /m	
	Subtotal b)	$67,500	
3.3.3	Assaying samples:		
	Number of samples	1,000	
	Cost per assay	$ 12.00 ea.	
	Subtotal c)	$12,000	
	Subsection Total (add a + b + c)		79,500

Reference Section No.	Item		Capital Cost $
3.4	Permits and Environmental Studies		
	Environmental sensitivity of region	Moderate	
	Are harmful contaminants produced?	Yes	
	Do contaminants require 'normal' or 'special' handling?	Normal	
	Subsection Total (enter cost from graph)		175,000
3.5	Project Management and Preproduction Scheduling		
	Average monthly cost	$ 9,500 /month	
	Duration of preproduction work	16 months	
	Subsection Total (multiply two lines above)		152,000
3.6	Access to Minesite		
3.6.1	New road construction: 5 km x $125,000/km =	$625,000	
3.6.2	Upgrading existing roads: ____ km x $____/km =	$ None	
3.6.3	Road Bridges: (total cost)	$ None	
3.6.4/7	Other Access Costs:	$ None	
	Subsection Total (add 4 lines above)		625,000
3.7	Site Preparation		

Site Category	Site Area m^2	Cost $	
A			
B	16,000	180,000	
C			
D			
Total			180,000

Reference Section No.	Item		Capital Cost $
3.8	Camp Installation		
	Total manpower 62 (see Form 2(b))		
	Camp capacity 69 personnel		
	Subsection Total (enter cost from graph)		395,000
3.9	Site Services		
	Ramp or shaft access Shaft		
	Production rate 260 t.p.d.		
	Subsection Total (enter cost from graph)		62,000
3.10	Electrical Power & Compressed Air		
3.10.1	Electrical Power		
	Site power requirements 1350 kW		
	Powerline:		
	Line Cost 5 km x $30,000/km = $150,000		
	Site Cost $200,000		
	Total Powerline Cost	$350,000	
	Generators: (enter cost from graph) =	$ N/A	
3.10.2	Compressor Plant		
	Compressed air requirements 1.60 m^3/sec.		
	Compressor inst'n. (enter cost from graph)	$160,000	
	Subsection Total (add 3 lines above)		510,000
3.11	Offices, Shops, Dry, Warehouse		
	Subsection Total (enter cost from graph)		189,000

Reference Section No.	Item		Capital Cost $
3.12	<u>Mine Access</u>		
	Complete 3.12.1, 3.12.2 or 3.12.3 below.		
3.12.1	<u>Shaft</u>		
	Shaft type 2-Compt. Timber		
	Shaft depth 330 m		
	Mobilize, setup, teardown, demobilize	$ 225,000	
	Shaft collar	$ 155,000	
	Shaft 315m x $ 4,600m =	$1,449,000	
	Shaft changeover to skipping =	$ 40,000	
	Total Shaft Costs (add 4 lines above)		1,869,000
3.12.2	<u>Decline</u>		
	Depth of lowest level ______ m		
	Mobilize, setup, teardown, demobilize	$_____	
	Decline portal	$_____	
	Decline excavation	$_____	
	Total Decline Costs (add 3 lines above)		N/A
3.12.3	<u>Adit(s)</u>		
	Mobilize, setup, teardown, demobilize	$_____	
	Adit portal	$_____	
	Adit excavation ____ m x $____/m =	$_____	
	Internal ramp (+15%) ____ m x $____/m =	$_____	
	Internal ramp (-15%) ____ m x $____/m =	$_____	
	Total Adit Costs (add 5 lines above)		N/A

Reference Section No.	Item		Capital Cost $
3.13	Ancillary Shaft Excavations & Installations		
3.13.1	Shaft stations: 7 x $63,500 ea. =	$444,500	
3.13.2	Loading pocket: =	$ 45,000	
3.13.3	Lip pockets: 3 x $15,000 ea. =	$ 45,000	
3.13.4	Spill handling: =	$ 5,000	
3.13.5	Shaft bottom construction: =	$ 10,000	
	Subsection Total (add 5 lines above)		$ 549,500
3.14	Hoisting System, Headframe & Bin		
	Hoisting depth 320 m		
	Hoisting capacity (ore & waste) 40 tonnes/hour		
3.14.1	Hoist & Hoistroom		
	Hoist selected:		
	Motor size 125 kw		
	Drum diameter 6 ft.		
	Total cost, hoist & hoistroom	$620,000	
3.14.2	Headframe & Collarhouse		
	Headframe height 26 m		
	Total cost, headframe & collarhouse	$265,000	
3.14.3	Headframe Bins		
	Bin or 'dump area' Dump area		
	If bin, what size? N/A tonnes		
	Total cost, bin or dump area	$ 15,000	
3.14.4	Conveyances		
	Conveyance combination Skip/Cage & Cwt.		
	Total cost, conveyances	$ 76,000	
	Subsection Total (add 4 lines above)		$ 976,000

Reference Section No.	Item	Capital Cost $
3.15	<u>Ventilation & Mine Air Heating</u>	
3.15.1	Primary ventilation fans $ 21,500	
3.15.2	Mine air heaters $ 26,500	
	Subsection Total (add 2 lines above)	$ 48,000
3.16	<u>Underground Development</u>	
3.16.1	<u>Level Development</u>	

Production level development costs are approximated by the following:

<u>Quantity of development per level</u>:

Avg. x-cut length 30 m + strike length 300 m = 330 (a)

Cost per metre $1,390 (b)

<u>Annual Production Tonnage</u>:

Multiply the two items below:

Daily production rate 260 t.p.d.

Working days per year 350 days

Annual production tonnage = 91,000 (c)

<u>Tonnes accessed per level</u>:

Multiply the four items below:

Strike length between mining limits 300 m

Average stoping width 3 m

Ore length between main levels 32 m

Ore tonnage factor 2.7 t/m³

Tonnes accessed = 77,760 (d)

Total Preproduction Level Development Cost

$$= \frac{(a) \times (b) \times (c) \times 2}{(d)}$$ $1,073,603

USE $1,073,600

Reference Section No.	Item		Capital Cost $
3.16.2	Ore Pass System		
	Distance between top & bottom production levels 180 m		
	Total cost, (enter cost from graph)	$285,000	
3.16.3	Primary Ventilation & Escapeway		
	Depth of bottom level below surface	300 m	
	Total cost, (enter cost from graph)	$480,000	
	Subsection Total (add 3 totals above)		1,838,600
3.17	Underground Installations		
3.17.1	Main Sumps and Pump Stations:		
	Is mine 'dry', 'average' or 'wet'? Dry	$ 57,500	
3.17.2	Rockbreaker & Grizzly:	$ 95,000	
3.17.3	Ore Pass Controls:		
	No. of controls 3 x $20,000/control	$ 60,000	
3.17.4	Underground Electrical Room/Load Centre:		
	No. of installations 3 x $37,000/inst'n.	$111,000	
3.17.5	Miscellaneous Installations:		
	No. of levels 7 x $25,000/level	$175,000	
	Subsection Total (add 5 lines above)		498,500
3.18	Equipment		
	Shaft, ramp or adit? Shaft		
	Track or trackless haulage? Track		
	Subsection Total (enter cost from graph)		$ 590,000

Reference Section No.	Item		Capital Cost $
3.19	Concentrator		
3.19.1	Concentrator construction:		
	Process type	Cyanide	
	Construction costs, (enter cost from graph)	$3,600,000	
3.19.2	Tailings disposal area:		
	Construction costs, (enter cost from graph)	$ 180,000	
	Subsection Total, (add 2 lines above)		3,780,000
3.20	Cost Contingency		
3.20.1	Contingency for items omitted	5 %	
3.20.2	Contingency for variations in conditions	5 %	
3.20.3	Contingency for delays due to location	3 %	
	Total Contingency Percentage, (add 3 lines above)		13 %

Transfer all subsection totals to the summary page.

FORM 3(b)

ONGOING CAPITAL COSTS

SUMMARY (costs are developed on the next two pages)

	Year 1	Year 2	Year 3	Year 4	Year 5
Ongoing Capital Development	308,951	308,951	308,951	30,895*	-
Exploration Development	61,790	61,790	61,790	61,790	61,790
Exploration Diamond Drilling	27,720	27,720	27,720	27,720	27,720
Equipment Replacement	14,750	38,350	59,000	88,500	26,550
Subtotal	413,211	436,811	457,461	208,905	116,060
Regional Cost Factor	1.16	1.16	1.16	1.16	1.16
TOTAL ONGOING CAPITAL COST	479,325	506,700	530,655	242,330	134,630

* Mine life is approximately 5 years. Because Preproduction Capital Costs allow for development of 2 year's stope tonnage, Ongoing Capital Development has been reduced to 10% in Year 4 and 0% in Year 5.

ONGOING CAPITAL COSTS - Detailed Calculation Form

Reference Section No.	Item	Capital Cost $/Year

3.21 Ongoing Capital Development (O.C.D.)

Calculation is similar to that used for preproduction level development, Form 3(a), Section 3.16.1, except that results are NOT multiplied by 2.

Ongoing Capital Development Costs can be approximated by the following:

Quantity of development per level 330 m

Cost/metre (Owner's cost) x $ 800/m (e)

Annual production rate x 91,000 tonnes/year

Tonnes accessed/level ÷ 77,760 tonnes/level

Total Ongoing Development Costs $308,951 (f)

3.22 Exploration Development

Enter 20% of Ongoing Capital Development above 61,790

3.23 Exploration Diamond Drilling

The annual cost of exploration diamond drilling is approximated by the following:

Quantity of drilling/setup:

No. of holes 4 x length/hole 50 = 200 (a)

Annual quantity of exploration development:

Total O.C.D. 308,951 (f) x 20% ÷ Cost/m 800 (e) = 77 (b)

Cost/metre drilled = 45.00 (c)

Setup interval = 25 (d)

Total Exploration Diamond Drilling Cost

$$= \frac{(a) \times (b) \times (c)}{(d)} = \$27,720$$

Reference Section No.	Item	Capital Cost $/Year
3.24	Equipment Replacement	

Total value of capital equipment $590,000
(From Form 3(a), Section 3.18)

Enter percentages from the graph and multiply by the total values above to estimate annual replacement costs by year.

	% of Total Value		Total Annual Replacement Costs
Year 1	2.5		14,750
Year 2	6.5		38,350
Year 3	10.0		59,000
Year 4	15.0		88,500
Year 5	18.0	x 25% *	26,550

* Because this is the last year of operation.

FORM 4

REGIONAL COST FACTORS

Project Location British Columbia - Use Blanket Factors

CAPITAL COST FACTOR

Cost Classification Area	Regional Cost Factor for ________ (From Table 4.3)		Percentage Distribution of Capital Costs (Determined by User)		Multiplication Product	
Labour	________	x	________%	=	____________	(a)
Plant & Equipment	________	x	________%	=	____________	(b)
Materials & Consumables	________	x	________%	=	____________	(c)
Hydro Power	________	x	________%	=	____________	(d)
Transportation	________	x	________%	=	____________	(e)
Provincial Tax	________	x	________%	=	____________	(f)
		TOTAL =	100 %			

CAPITAL COST REGIONAL COST FACTOR (sum a to f) = ____________

OPERATING COST FACTOR

Cost Classification Area	Regional Cost Factor for ________ (From Table 4.3)		Percentage Distribution of Operating Costs (Determined by User)		Multiplication Product	
Labour	________	x	________%	=	____________	(g)
Plant & Equipment	________	x	________%	=	____________	(h)
Materials & Consumables	________	x	________%	=	____________	(i)
Hydro Power	________	x	________%	=	____________	(j)
Transportation	________	x	________%	=	____________	(k)
Provincial Tax	________	x	________%	=	____________	(l)
		TOTAL =	100 %			

OPERATING COST REGIONAL COST FACTOR (sum g to l) = ____________

FORM 5(a)

GEOLOGICAL TONNES & GRADE

SUMMARY - From Forms 5(b)		Unit of Measurement
Primary metal is	Gold (Au)	g/tonne
Secondary metal is	Silver (Ag)	g/tonne
Tertiary metal is	N/A	N/A

Cross Section No.	Total Tonnes of Mineralization	Total Units of Metal		
		Primary Metal	Secondary Metal	Tertiary Metal
0 + 00E	43,875	667,778	2,874,866	-
0 + 50E	72,900	890,676	3,946,725	-
0 + 100E	100,575	1,026,189	4,327,020	-
0 + 150E	88,425	903,487	3,880,919	-
0 + 200E	70,200	856,615	3,291,327	-
0 + 250E	105,367	1,166,371	5,987,728	-
	(A)	(B)	(C)	(D)
TOTALS	481,342	5,511,116	24,308,585	N/A

TOTAL GEOLOGICAL RESERVE = 481,342 tonnes (A)

GEOLOGICAL RESERVE GRADE:

Primary Metal (B/A) = 11.45 g/tonne

Secondary Metal (C/A) = 50.50 g/tonne

Tertiary Metal (D/A) = N/A

FORM 5(b)

GEOLOGICAL TONNES & GRADE

MINERAL RESERVE BY CROSS-SECTION

Cross-Section No. 0 + 00

	Mineralized Area			
	A	B	C	D
Diamond Drill Hole No.				
Area of Mineralization (m^2)		195		130
Grade of Mineralized Area:				
Primary Metal		17.14		12.34
Secondary Metal		68.24		61.45
Tertiary Metal		-		-
Strike Length Represented by Cross-section (m)		50		50
Volume of Mineralization (m^3)		9,750		6,500
Tonnage Factor (tonnes/m^3)		2.7		2.7
Tonnes of Mineralization		26,325		17,550
Total Units of Metal:				
Primary Metal		451,211		216,567
Secondary Metal		1,796,418		1,078,448
Tertiary Metal				

TOTAL TONNES (A + B + C + D) = 43,875

	Total Units of Metal	Average Grade
Primary Metal	667,778 g	15.22 g/tonne
Secondary Metal	2,874,866 g	65.52 g/tonne
Tertiary Metal		

FORM 5(b)

GEOLOGICAL TONNES & GRADE

MINERAL RESERVE BY CROSS-SECTION

Cross-Section No. 0 + 50

	Mineralized Area			
	A	B	C	D
Diamond Drill Hole No.				
Area of Mineralization (m^2)	330		210	
Grade of Mineralized Area:				
Primary Metal	12.42		11.90	
Secondary Metal	50.60		59.70	
Tertiary Metal	-		-	
Strike Length Represented by Cross-section (m)	50		50	
Volume of Mineralization (m^3)	16,500		10,500	
Tonnage Factor (tonnes/m^3)	2.7		2.7	
Tonnes of Mineralization	44,550		28,350	
Total Units of Metal:				
Primary Metal	553,311		337,365	
Secondary Metal	2,254,230		1,692,495	
Tertiary Metal	-		-	

TOTAL TONNES (A + B + C + D) = 72,900

	Total Units of Metal	Average Grade
Primary Metal	890,676	12.22
Secondary Metal	3,946,725	54.14
Tertiary Metal		

FORM 5(b)

GEOLOGICAL TONNES & GRADE

MINERAL RESERVE BY CROSS-SECTION

Cross-Section No. 0 + 100

	Mineralized Area			
	A	B	C	D
Diamond Drill Hole No.				
Area of Mineralization (m^2)	445		300	
Grade of Mineralized Area:				
Primary Metal	9.72		10.92	
Secondary Metal	45.6		39.2	
Tertiary Metal	-		-	
Strike Length Represented by Cross-section (m)	50		50	
Volume of Mineralization (m^3)	22,250		15,000	
Tonnage Factor (tonnes/m^3)	2.7		2.7	
Tonnes of Mineralization	60,075		40,500	
Total Units of Metal:				
Primary Metal	583,929		442,260	
Secondary Metal	2,739,420		1,587,600	
Tertiary Metal	-		-	

TOTAL TONNES (A + B + C + D) = 100,575

	Total Units of Metal	Average Grade
Primary Metal	1,026,189	10.20
Secondary Metal	4,327,020	43.02
Tertiary Metal		

FORM 5(b)

GEOLOGICAL TONNES & GRADE

MINERAL RESERVE BY CROSS-SECTION

Cross-Section No. 0 + 150

	Mineralized Area			
	A	B	C	D
Diamond Drill Hole No.				
Area of Mineralization (m^2)		390	265	
Grade of Mineralized Area:				
Primary Metal		9.55	11.20	
Secondary Metal		45.71	41.21	
Tertiary Metal		-	-	
Strike Length Represented by Cross-section (m)		50	50	
Volume of Mineralization (m^3)		19,500	13,250	
Tonnage Factor (tonnes/m^3)		2.7	2.7	
Tonnes of Mineralization		52,650	35,775	
Total Units of Metal:				
Primary Metal		502,807	400,680	
Secondary Metal		2,406,631	1,474,288	
Tertiary Metal		-	-	

TOTAL TONNES (A + B + C + D) = 88,425

	Total Units of Metal	Average Grade
Primary Metal	903,487	10.22
Secondary Metal	3,880,919	43.89
Tertiary Metal		

FORM 5(b)

GEOLOGICAL TONNES & GRADE

MINERAL RESERVE BY CROSS-SECTION

Cross-Section No. 0 + 200

	Mineralized Area			
	A	B	C	D
Diamond Drill Hole No.				
Area of Mineralization (m^2)		325	195	
Grade of Mineralized Area:				
Primary Metal		13.32	10.34	
Secondary Metal		51.07	39.91	
Tertiary Metal		-	-	
Strike Length Represented by Cross-section (m)		50	50	
Volume of Mineralization (m^3)		16,250	9,750	
Tonnage Factor (tonnes/m^3)		2.7	2.7	
Tonnes of Mineralization		43,875	26,325	
Total Units of Metal:				
Primary Metal		584,415	272,200	
Secondary Metal		2,240,696	1,050,631	
Tertiary Metal		-	-	

TOTAL TONNES (A + B + C + D) = 70,200

	Total Units of Metal	Average Grade
Primary Metal	856,615	12.20
Secondary Metal	3,291,327	46.88
Tertiary Metal	-	-

FORM 5(b)

GEOLOGICAL TONNES & GRADE

MINERAL RESERVE BY CROSS-SECTION

Cross-Section No. 0 + 250

	Mineralized Area			
	A	B	C	D
Diamond Drill Hole No.				
Area of Mineralization (m^2)		446.5		334
Grade of Mineralized Area:				
Primary Metal		9.85		12.70
Secondary Metal		39.74		79.67
Tertiary Metal		-		-
Strike Length Represented by Cross-section (m)		50		50
Volume of Mineralization (m^3)		22,325		16,700
Tonnage Factor (tonnes/m^3)		2.7		2.7
Tonnes of Mineralization		60,277		45,090
Total Units of Metal:				
Primary Metal		593,728		572,643
Secondary Metal		2,395,408		3,592,320
Tertiary Metal		-	-	

TOTAL TONNES (A + B + C + D) = 105,367

	Total Units of Metal	Average Grade
Primary Metal	1,166,371	11.07
Secondary Metal	5,987,728	56.83
Tertiary Metal	-	-

FORM 5(c)

IN-SITU TONNES & GRADE

SUMMARY - From Forms 5(d)

		Unit of Measurement
Primary metal is	Gold (Au)	g/tonne
Secondary metal is	Silver (Ag)	g/tonne
Tertiary metal is	N/A	N/A

Cross Section No.	Total In-Situ Tonnes	Total Units of Metal		
		Primary Metal	Secondary Metal	Tertiary Metal
0 + 00E	49,410	667,778	2,874,866	
0 + 50E	76,005	841,217	3,727,053	
0 + 100E	105,300	971,271	4,093,740	
0 + 150E	91,800	848,576	3,643,197	
0 + 200E	75,195	813,885	3,127,137	
0 + 250E	111,577	1,166,371	5,987,728	
	(A)	(B)	(C)	(D)
TOTALS	509,287	5,309,098	23,453,721	

TOTAL IN-SITU RESERVE = 509,287 (A)

IN-SITU RESERVE GRADE:

Primary Metal (B/A) = 10.42 g/tonne

Secondary Metal (C/A) = 46.05 g/tonne

Tertiary Metal (D/A) =

FORM 5(d)

IN-SITU TONNES & GRADE

IN-SITU RESERVE BY CROSS-SECTION

Cross-Section No. 0 + 00

	Mining Zone			
	A	B	C	D
AREAS (Within Mining Limits)				
Area of Mineralization (m^2)		195		130
Area of Waste (m^2)		23		18
Total Area (m^2)		218		148
VOLUMES (Within Mining Limits)				
Strike Length Represented by Cross-section (m)		50		50
Volume of Mineralization (m^3)		9,750		6,500
Volume of Waste (m^3)		1,150		900
Total Volume (m^3)		10,900		7,400
TONNES (Within Mining Limits)				
Tonnage Factor - Ore (tonnes/m^3)		2.7		2.7
Tonnage Factor - Waste (tonnes/m^3)		2.7		2.7
Tonnes of Mineralization		26,325		17,550
Tonnes of Waste		3,105		2,430
Tonnes of Mineralization & Waste (Ore)		29,430		19,980
TOTAL TONNES OF ORE - ALL ZONES		49,410 (a)		
GRADES OF MINERALIZED AREAS (Geological Grade)				
Primary Metal		17.14		12.34
Secondary Metal		68.24		61.45
Tertiary Metal		-		
UNITS OF METAL (Tonnes of Mineralization x Geological Grade)				
Primary Metal		451,211		216,567
Secondary Metal		1,796,418		1,078,448
Tertiary Metal		-		-

	Total Units All Zones (b)	Average Grade All Zones (within mining limits) (b) ÷ (a)
Primary	667,778	13.52
Secondary	2,874,866	58.18
Tertiary		

FORM 5(d)

IN-SITU TONNES & GRADE

IN-SITU RESERVE BY CROSS-SECTION

Cross-Section No. 0 + 50

	Mining Zone			
	A	B	C	D
AREAS (Within Mining Limits)				
Area of Mineralization (m^2)	312		198	
Area of Waste (m^2)	32		21	
Total Area (m^2)	344		219	
VOLUMES (Within Mining Limits)				
Strike Length Represented by Cross-section (m)	50		50	
Volume of Mineralization (m^3)	15,600		9,900	
Volume of Waste (m^3)	1,600		1,050	
Total Volume (m^3)	17,200		10,950	
TONNES (Within Mining Limits)				
Tonnage Factor - Ore (tonnes/m^3)	2.7		2.7	
Tonnage Factor - Waste (tonnes/m^3)	2.7		2.7	
Tonnes of Mineralization	42,120		26,730	
Tonnes of Waste	4,320		2,835	
Tonnes of Mineralization & Waste (Ore)	46,440		29,565	
TOTAL TONNES OF ORE - ALL ZONES		76,005 (a)		
GRADES OF MINERALIZED AREAS (Geological Grade)				
Primary Metal	12.42		11.9	
Secondary Metal	50.60		59.7	
Tertiary Metal	-			
UNITS OF METAL (Tonnes of Mineralization x Geological Grade)				
Primary Metal	523,130		318,087	
Secondary Metal	2,131,272		1,595,781	
Tertiary Metal	-		-	

	Total Units All Zones (b)	Average Grade All Zones (within mining limits) (b) ÷ (a)
Primary	841,217	11.07
Secondary	3,727,053	49.04
Tertiary		

FORM 5(d)

IN-SITU TONNES & GRADE

IN-SITU RESERVE BY CROSS-SECTION

Cross-Section No. 0 + 100

		Mining Zone		
	A	B	C	D
AREAS (Within Mining Limits)				
Area of Mineralization (m^2)	420		285	
Area of Waste (m^2)	45		30	
Total Area (m^2)	465		315	
VOLUMES (Within Mining Limits)				
Strike Length Represented by Cross-section (m)	50		50	
Volume of Mineralization (m^3)	21,000		14,250	
Volume of Waste (m^3)	2,250		1,500	
Total Volume (m^3)	23,250		15,750	
TONNES (Within Mining Limits)				
Tonnage Factor - Ore (tonnes/m^3)	2.7		2.7	
Tonnage Factor - Waste (tonnes/m^3)	2.7		2.7	
Tonnes of Mineralization	56,700		38,475	
Tonnes of Waste	6,075		4,050	
Tonnes of Mineralization & Waste (Ore)	62,775		42,525	
TOTAL TONNES OF ORE - ALL ZONES		105,300 (a)		
GRADES OF MINERALIZED AREAS (Geological Grade)				
Primary Metal	9.72		10.92	
Secondary Metal	45.6		39.20	
Tertiary Metal	-		-	
UNITS OF METAL (Tonnes of Mineralization x Geological Grade)				
Primary Metal	551,124		420,147	
Secondary Metal	2,585,520		1,508,220	
Tertiary Metal	-		-	

	Total Units All Zones (b)	Average Grade All Zones (within mining limits) (b) ÷ (a)
Primary	971,271	9.22
Secondary	4,093,740	38.88
Tertiary	-	-

FORM 5(d)

IN-SITU TONNES & GRADE

IN-SITU RESERVE BY CROSS-SECTION

Cross-Section No. 0 + 150

		Mining Zone		
	A	B	C	D
AREAS (Within Mining Limits)				
Area of Mineralization (m^2)		365	250	
Area of Waste (m^2)		40	25	
Total Area (m^2)		405	275	
VOLUMES (Within Mining Limits)				
Strike Length Represented by Cross-section (m)		50	50	
Volume of Mineralization (m^3)		18,250	12,500	
Volume of Waste (m^3)		2,000	1,250	
Total Volume (m^3)		20,250	13,750	
TONNES (Within Mining Limits)				
Tonnage Factor - Ore (tonnes/m^3)		2.7	2.7	
Tonnage Factor - Waste (tonnes/m^3)		2.7	2.7	
Tonnes of Mineralization		49,275	33,750	
Tonnes of Waste		5,400	3,375	
Tonnes of Mineralization & Waste (Ore)		54,675	37,125	
TOTAL TONNES OF ORE - ALL ZONES		91,800 (a)		
GRADES OF MINERALIZED AREAS (Geological Grade)				
Primary Metal		9.55	11.20	
Secondary Metal		45.71	41.21	
Tertiary Metal		-	-	
UNITS OF METAL (Tonnes of Mineralization x Geological Grade)				
Primary Metal		470,576	378,000	
Secondary Metal		2,252,360	1,390,837	
Tertiary Metal		-	-	

	Total Units All Zones (b)	Average Grade All Zones (b) ÷ (a) (within mining limits)
Primary	848,576	9.24
Secondary	3,643,197	39.69
Tertiary		

FORM 5(d)

IN-SITU TONNES & GRADE

IN-SITU RESERVE BY CROSS-SECTION

Cross-Section No. 0 + 200

		Mining Zone		
	A	B	C	D
AREAS (Within Mining Limits)				
Area of Mineralization (m^2)		309	185	
Area of Waste (m^2)		36	27	
Total Area (m^2)		345	212	
VOLUMES (Within Mining Limits)				
Strike Length Represented by Cross-section (m)		50	50	
Volume of Mineralization (m^3)		15,450	9,250	
Volume of Waste (m^3)		1,800	1,350	
Total Volume (m^3)		17,250	10,600	
TONNES (Within Mining Limits)				
Tonnage Factor - Ore (tonnes/m^3)		2.7	2.7	
Tonnage Factor - Waste (tonnes/m^3)		2.7	2.7	
Tonnes of Mineralization		41,715	24,975	
Tonnes of Waste		4,860	3,645	
Tonnes of Mineralization & Waste (Ore)		46,575	28,620	
TOTAL TONNES OF ORE - ALL ZONES		75,195 (a)		
GRADES OF MINERALIZED AREAS (Geological Grade)				
Primary Metal		13.32	10.34	
Secondary Metal		51.07	39.91	
Tertiary Metal		-	-	
UNITS OF METAL (Tonnes of Mineralization x Geological Grade)				
Primary Metal		555,644	258,241	
Secondary Metal		2,130,385	996,752	
Tertiary Metal		-	-	

	Total Units All Zones (b)	Average Grade All Zones (b) ÷ (a) (within mining limits)
Primary	813,885	10.82
Secondary	3,127,137	41.59
Tertiary		

FORM 5(d)

IN-SITU TONNES & GRADE

IN-SITU RESERVE BY CROSS-SECTION

Cross-Section No. 0 + 250

		Mining Zone		
	A	B	C	D
AREAS (Within Mining Limits)				
Area of Mineralization (m²)		446.5		334
Area of Waste (m²)		25.00		21
Total Area (m²)		471.50		355
VOLUMES (Within Mining Limits)				
Strike Length Represented by Cross-section (m)		50		50
Volume of Mineralization (m³)		22,325		16,700
Volume of Waste (m³)		1,250		1,050
Total Volume (m³)		23,575		17,750
TONNES (Within Mining Limits)				
Tonnage Factor - Ore (tonnes/m³)		2.7		2.7
Tonnage Factor - Waste (tonnes/m³)		2.7		2.7
Tonnes of Mineralization		60,277		45,090
Tonnes of Waste		3,375		2,835
Tonnes of Mineralization & Waste (Ore)		63,652		47,925
TOTAL TONNES OF ORE - ALL ZONES		111,577 (a)		
GRADES OF MINERALIZED AREAS (Geological Grade)				
Primary Metal		9.85		12.70
Secondary Metal		39.74		79.67
Tertiary Metal		-		-
UNITS OF METAL (Tonnes of Mineralization x Geological Grade)				
Primary Metal		593,728		572,643
Secondary Metal		2,395,408		3,592,320
Tertiary Metal		-		-

	Total Units All Zones (b)	Average Grade All Zones (b) ÷ (a) (within mining limits)
Primary	1,166,371	10.45
Secondary	5,987,728	53.66
Tertiary		

FORM 5(e)

MINEABLE TONNES AND GRADE TO MILL

ADJUSTMENT FOR MINING RECOVERY

			Total In-Situ Reserve		Mining Recovery Factor		Actually Mined	
Tonnes		=	509,287	x	0.90	=	458,358	(a)
Primary Metal	- Units	=	5,309,098 g	x	0.90	=	4,778,188 g	(b)
	- Grade	=	10.42 g/t				10.42 g/t	(c)
Secondary Metal	- Units	=	23,453,721 g	x	0.90	=	21,108,349 g	(d)
	- Grade	=	46.05 g/t				46.05 g/t	(e)
Tertiary Metal	- Units	=	N/A	x		=	N/A	(f)
	- Grade	=						(g)

ADJUSTMENT FOR DILUTION

Dilution Factor = 1.15 (h)

Diluted Tonnes (a x h) = 527,112 (i)

"MINEABLE GRADES"

Primary Metal (c/h) = 9.06 g/t (j)
Secondary Metal (e/h) = 40.04 g/t (k)
Tertiary Metal (g/h) = N/A (l)

ADJUSTMENT FOR STOPE LOSSES

Stope Losses Factor = 0.95 (m)

"MINEABLE TONNES" to Mill (i x m) = 500,756 (n)

Units of Metal/Tonne in Mill Feed:

Primary Metal (b/i) = 9.06 g/t
Secondary Metal (d/i) = 40.05 g/t
Tertiary Metal (f/i) = N/A

FORM 5(f)

MINERAL DEPOSIT VALUE

	Primary Metal	Secondary Metal	Tertiary Metal
Units of Metal/Tonne in Mill Feed	9.06 g	40.05 g	N/A
Mill Recovery Factor	0.95	0.85	
Units of Metal Recovered Per Tonne of Mill Feed	8.61 g	34.04 g	
Current Metal Price/Unit	$15.11/g ($470 Cdn/oz)	$0.23/g ($7.15 Cdn/oz)	
Value Per Tonne of Ore after Milling	$130.10	$7.83	

TOTAL VALUE PER TONNE OF ORE AFTER MILLING = $137.93

NET VALUE PER TONNE OF ORE AFTER SMELTING/REFINING (Approximation only) = $130.00

FORM 6

PRELIMINARY CASH FLOW SUMMARY

Reference Section No.	Cash Flow (Initial 5 years)	$ in 1,000's					
	Year	0	1	2	3	4	5
Form 3(a)	Preproduction Capital	16,742					
Form 3(b)	Ongoing Capital		479	507	531	242	135
	Total Capital	16,742	479	507	531	242	135
Form 3(a) Section 3.16.1	Annual Production 91,000 tonnes						
Form 2(a)	Operating Cost/Tonne $100.53/tonne						
	Annual Operating Costs		9,148	9,148	9,148	9,148	9,148
	Total On-Site Costs	16,742	9,627	9,655	9,679	9,390	9,283
Form 2(a) Section 2.14	Freight Charges $ N/A /tonne of Ore						
	Annual Freight Costs			-	-	-	-
	Total Costs	16,742	9,627	9,655	9,679	9,390	9,283
Form 5(f)	Revenue/Tonne after Smelting $130.00/tonne						
	Total Revenue		11,830	11,830	11,830	11,830	11,830
	* Cash Flow - By Year	-16,742	2,203	2,175	2,151	2,440	2,547
	* Cumulative Cash Flow	-16,742	-14,539	-12,364	-10,213	-7,773	-5,226

* Before depreciation and taxes.

EXAMPLE TWO

FORM 1

BASIC INFORMATION

Estimate Prepared By: Henry Brown Date: June 13, 1986

Name of Property: PROFIT GOLD MINE

Property Location: 10 km outside of Val d'Or, Quebec

Brief Description of Site and Local Area: Uneven, dry area with rock outcrops

Expected Overburden Conditions (Depth and Type): Minimal

Expected Rock Conditions: Competent ore and wall rocks

Expected Ground Water Conditions: Moderate

Other Relevant Information: For the purpose of this example, the following tonnages and grades are assumed:

Geological tonnes and grade: 900,000 tonnes @ 11 g/tonne Au

In-situ tonnes and grade: 850,000 tonnes @ 10 g /tonne Au

Ore horizon is from 50 to 150 metres below surface. The deposit measures 550 metres long, 6 m wide and 100 metres high

Attach additional sheets as required.

FORM 2(a)

OPERATING COSTS

SUMMARY (costs are developed on the next two pages)

		$/Tonne	
Stoping Costs		10.50	
Hoisting or Ramp Haulage Cost		2.15	
Level Haulage Cost		3.10	
General Mine Expense		4.85	
Surface Plant and Mine Services		8.61	
Staff and Administration		6.00	
Milling		20.00	
	Subtotal	55.21	
Add Cost Contingency @ 11%		6.07	
	Subtotal	61.28	(a)
Regional Operating Cost Factor	(blanket factor)	1.05	(b)
TOTAL OPERATING COST (a) x (b)		64.34	
Transportation of Mine Product		2.00	

FORM 2(a)

OPERATING COSTS

DETAILED CALCULATION FORM

Reference Section No.	Item	Operating Cost $/Tonne Ore
2.2	Selection of Production Rate	
	Rate selected 417 tonnes/day	
	Mining days per year 350	
	Mining shifts per day 3	
2.3	Selection of Mining Method	
	Method selected Blasthole	
2.4	Stoping Costs	10.50
2.5	Selection of Mine Access and Haulage Method	
	Access selected Ramp	
	Level haulage selected Trackless	
	Depth 75 m (avg.)	
2.6	Hoisting or Ramp Haulage Cost	2.15
2.7	Level Haulage Cost	
	Haulage Distance 150 m (avg.)	
	Haulage Capacity/Trip 2 yd.^{3}	
	Cost	3.10
2.8	General Mine Expense	4.85

2.9 Surface Plant and Mine Services

Power source selected Hydro

Cost

a)	Labour	3.40	
b)	Materials and Operating Costs	1.35	
c)	Power	2.25	
d)	Camp (50% in camp)	1.60	
e)	Road Maintenance	0.01	
	Total Surface Plant and Mine Services		8.61

2.10 Staff and Administration

6.00

2.11 Milling

Selection of Location Custom - Val d'Or

Cost 20.00

2.14 Transportation of Mine Product

i)	Bullion	N/A
ii)	Ore (enter cost from graph)	2.00

iii) Concentrate ______ $/tonne (a)
(enter cost from graph)

Concentrating Ratio ______ (b)

Cost/tonne mined (a) ÷ (b) N/A

3.20 Cost Contingency

3.20.1	Contingency for items omitted	5 %	
3.20.2	Contingency for variations in conditions	5 %	
3.20.3	Contingency for delays due to location	1 %	
	Total Contingency Percentage, (add 3 lines above)		11 %

Transfer all subsection totals to the summary page.

FORM 2(b)

MANPOWER SCHEDULE

Reference Section No.	Item			Manpower
3.21	Ongoing Capital Development			1
2.4	Stoping			
	Mining method selected	Blasthole		
	Productivity (tonnes/manshift)	47	(a)	
	Production rate (tonnes/day)	417	(b)	
	Manpower required (b)/(a)			9
2.6	Hoisting or Ramp Haulage			
	Hoisting:			
	Shifts worked per day		(c)	
	Manpower required per shift	2	(d)	
	Manpower required per day (c) x (d)			N/A
	Ramp:			
	Vertical depth	75 m	(e)	
	Manpower required			3
2.7	Level Haulage			
	Haulage method selected	2-yd.^{3} LHD		
	Manpower required			4
2.8	General Mine Expense			
	Track or trackless mine	Trackless		10
	Subtotal Underground Manpower (including hoistman)			27
2.9	Surface Plant and Mine Services			10
2.10	Staff and Administration			11
2.11	Milling			N/A
	Subtotal Surface Manpower			21
	TOTAL ON-SITE MANPOWER			48

FORM 3(a)

PREPRODUCTION CAPITAL COSTS

SUMMARY (costs are developed on the next 8 pages)	$	
Feasibility Studies and Detailed Engineering	205,000	
Additional Diamond Drilling and Sampling	338,250	
Permits and Environmental Studies	175,000	
Project Management and Preproduction Scheduling	114,000	
Access to Minesite	250,000	
Site Preparation	165,000	
Camp Installation	165,000	
Site Services	92,000	
Electrical Power & Compressed Air	495,000	
Offices, Shops, Dry, Warehouse	218,000	
Mine Access	2,190,000	
Ancillary Shaft Excavations & Installations	N/A	
Hoisting Systems, Headframes & Bins	N/A	
Ventilation & Mine Air Heating	70,000	
Underground Development	786,000	
Underground Installations	214,000	
Equipment	1,150,000	
Concentrator	N/A	
Subtotal	6,627,250	
Add Cost Contingency @ 11 %	728,998	
Subtotal	7,356,248	(a)
Regional Capital Cost Factor (blanket factor)	1.05	(b)
TOTAL PREPRODUCTION CAPITAL COST (a) x (b)	7,724,060	
USE	7,724,000	

PREPRODUCTION CAPITAL COSTS - Detailed Calculation Form

Reference Section No.	Item		Capital Cost $
3.2	Feasibility Studies & Detailed Engineering		
	For shaft or ramp access?	Ramp	
	Subsection Total (enter cost from graph)		205,000
3.3	Additional Diamond Drilling & Sampling		
3.3.1	Drilling from surface:		
	Number of holes	18	
	Average hole length	175 m	
	Cost/metre	$ 65.00 /m	
	Subtotal a)	$204,750	
3.3.2	Underground drilling:		
	Number of holes	54	
	Average hole length	50 m	
	Cost/metre	$ 45.00 /m	
	Subtotal b)	$121,500	
3.3.3	Assaying samples:		
	Number of samples	1,000	
	Cost per assay	$ 12.00 ea.	
	Subtotal c)	$ 12,000	
	Subsection Total (add a + b + c)		338,250

Reference Section No.	Item		Capital Cost $
3.4	Permits and Environmental Studies		
	Environmental sensitivity of region	Moderate	
	Are harmful contaminants produced?	Yes	
	Do contaminants require 'normal' or 'special' handling?	Normal	
	Subsection Total (enter cost from graph)		175,000
3.5	Project Management and Preproduction Scheduling		
	Average monthly cost	$ 9,500 /month	
	Duration of preproduction work	12 months	
	Subsection Total (multiply two lines above)		114,000
3.6	Access to Minesite		
3.6.1	New road construction: 2 km x $100,000/km =	$200,000	
3.6.2	Upgrading existing roads: 2 km x $ 25,000/km =	$ 50,000	
3.6.3	Road bridges: (total cost)	$ N/A	
3.6.4/7	Other Access Costs:	$ N/A	
	Subsection Total (add 4 lines above)		250,000
3.7	Site Preparation		

Site Category	Site Area m^2	Cost $	
A			
B	14,500	165,000	
C			
D			
Total			165,000

Reference Section No.	Item		Capital Cost $
3.8	Camp Installation		
	Total manpower 48 (see Form 2(b))		
	Camp capacity 27 personnel		
	Subsection Total (enter cost from graph)		165,000
3.9	Site Services		
	Ramp or shaft access Ramp		
	Production rate 417 t.p.d.		
	Subsection Total (enter cost from graph)		92,000
3.10	Electrical Power & Compressed Air		
3.10.1	Electrical Power		
	Site power requirements 1200 kW		
	Powerline:		
	Line Cost 4 km x $30,000/km = $120,000		
	Site Cost $200,000		
	Total Powerline Cost	$320,000	
	Generators: (enter cost from graph) =	$ N/A	
3.10.2	Compressor Plant		
	Compressed air requirements 1.85 m^3/sec.		
	Compressor inst'n. (enter cost from graph)	$175,000	
	Subsection Total (add 3 lines above)		495,000
3.11	Offices, Shops, Dry, Warehouse		
	Subsection Total (enter cost from graph)		218,000

Reference Section No.	Item		Capital Cost $
3.12	Mine Access		
	Complete 3.12.1, 3.12.2, or 3.12.3 below.		
3.12.1	Shaft		
	Shaft type ____________		
	Shaft depth ________ m		
	Mobilize, setup, teardown, demobilize	$________	
	Shaft collar	$________	
	Shaft _____ m x $______/m =	$________	
	Shaft changeover to skipping =	$________	
	Total Shaft Costs (add 4 lines above)		N/A
3.12.2	Decline		
	Depth of lowest level 150 m		
	Mobilize, setup, teardown, demobilize	$ 160,000	
	Decline portal	$ 105,000	
	Decline excavation	$1,925,000	
	Total Decline Costs (add 3 lines above)		2,190,000
3.12.3	Adit(s)		
	Mobilize, setup, teardown, demobilize	$______	
	Adit portal	$______	
	Adit excavation ____ m x $_____/m =	$______	
	Internal ramp (+15%) ____ m x $_____/m =	$______	
	Internal ramp (-15%) ____ m x $_____/m =	$______	
	Total Adit Costs (add 5 lines above)		N/A

Reference Section No.	Item		Capital Cost $
3.13	Ancillary Shaft Excavations & Installations		
3.13.1	Shaft stations: ____ x $ ____ ea. =	$ ____	
3.13.2	Loading pocket: =	$ ____	
3.13.3	Lip pockets: ____ x $ ____ ea. =	$ ____	
3.13.4	Spill handling: =	$ ____	
3.13.5	Shaft bottom construction: =	$ ____	
	Subsection Total (add 5 lines above)		$ N/A
3.14	Hoisting System, Headframe & Bin		
	Hoisting depth ____ m		
	Hoisting capacity (ore & waste) ____ tonnes/hour		
3.14.1	Hoist & Hoistroom		
	Hoist selected:		
	Motor size ____ kw		
	Drum diameter ____ ft.		
	Total cost, hoist & hoistroom	$ ____	
3.14.2	Headframe & Collarhouse		
	Headframe height ____ m		
	Total cost, headframe & collarhouse	$ ____	
3.14.3	Headframe Bins		
	Bin or 'dump area' ____		
	If bin, what size? ____ tonnes		
	Total cost, bin or dump area	$ ____	
3.14.4	Conveyances		
	Conveyance combination ____		
	Total cost, conveyances	$ ____	
	Subsection Total (add 4 lines above)		$ N/A

Reference Section No.	Item					Capital Cost $
3.15	Ventilation & Mine Air Heating					
3.15.1	Primary ventilation fans	$ 32,000				
3.15.2	Mine air heaters	$ 38,000				
	Subsection Total (add 2 lines above)					$ 70,000
3.16	Underground Development					
3.16.1	Level Development					
	Production level development costs are approximated by the following:					
	Quantity of development per level:					
	Avg. x-cut length 15 m + strike length 550 m =			565	(a)	
	Cost per metre			$1,475	(b)	
	Annual Production Tonnage:					
	Multiply the two items below:					
	Daily production rate	417	t.p.d.			
	Working days per year	350	days			
	Annual production tonnage =			145,950	(c)	
	Tonnes accessed per level:					
	Multiply the four items below:					
	Strike length between mining limits	550	m			
	Average stoping width	6	m			
	Ore length between main levels	50	m			
	Ore tonnage factor	2.7	t/m^3			
	Tonnes accessed =			445,500	(d)	
	Total Preproduction Level Development Cost $= \frac{(a) \times (b) \times (c) \times 2}{(d)}$			$ 546,043		
	USE			$ 546,000		

Reference Section No.	Item	Capital Cost $
3.16.2	<u>Ore Pass System</u>	
	Distance between top & bottom production levels ____ m	
	Total cost, (enter cost from graph) $ N/A	
3.16.3	<u>Primary Ventilation & Escapeway</u>	
	Depth of bottom level below surface 150 m	
	Total cost, (enter cost from graph) $240,000	
	Subsection Total (add 3 totals above)	786,000
3.17	<u>Underground Installations</u>	
3.17.1	Main Sumps and Pump Stations:	
	Is mine 'dry', 'average' or 'wet'? Avg. $ 65,000	
3.17.2	Rockbreaker & Grizzly: $ N/A	
3.17.3	Ore Pass Controls:	
	No. of controls ____ x $ ____/control $ N/A	
3.17.4	Underground Electrical Room/Load Centre:	
	No. of installations 2 x $37,000/inst'n. $ 74,000	
3.17.5	Miscellaneous Installations:	
	No. of levels 3 x $25,000/level $ 75,000	
	Subsection Total (add 5 lines above)	214,000
3.18	<u>Equipment</u>	
	Shaft, ramp or adit? Ramp	
	Track or trackless haulage? Trackless	
	Subsection Total (enter cost from graph)	$1,150,000

Reference Section No.	Item		Capital Cost $
3.19	<u>Concentrator</u>		
3.19.1	Concentrator construction:		
	Process type ______		
	Construction costs, (enter cost from graph)	$ ________	
3.19.2	Tailings disposal area:		
	Construction costs, (enter cost from graph)	$ ________	
	Subsection Total, (add 2 lines above)		N/A
3.20	<u>Cost Contingency</u>		
3.20.1	Contingency for items omitted	5 %	
3.20.2	Contingency for variations in conditions	5 %	
		1	
3.20.3	Contingency for delays due to location	____ %	
	Total Contingency Percentage, (add 3 lines above)		11 %

Transfer all subsection totals to the summary page.

FORM 3(b)

ONGOING CAPITAL COSTS

SUMMARY (costs are developed on the next two pages)

	Year 1	Year 2	Year 3	Year 4	Year 5
Ongoing Capital Development	175,844	175,844	175,844	175,844	17,584*
Exploration Development	35,169	35,169	35,169	35,169	35,169
Exploration Diamond Drilling	15,984	15,984	15,984	15,984	15,984
Equipment Replacement	28,750	74,750	115,000	172,500	207,000
Subtotal	255,747	301,747	341,997	399,497	275,737
Regional Cost Factor	1.05	1.05	1.05	1.05	1.05
TOTAL ONGOING CAPITAL COST	268,534	316,834	359,097	419,472	289,524

* Mine life is approximately 6 years. Ongoing Capital Development has been reduced to 10% in Year 5.

ONGOING CAPITAL COSTS - Detailed Calculation Form

Reference Section No. — Item — Capital Cost $/Year

3.21 Ongoing Capital Development (O.C.D.)

Calculation is similar to that used for preproduction level development, Form 3(a), Section 3.16.1, except that results are NOT multiplied by 2.

Ongoing Capital Development Costs can be approximated by the following:

Quantity of development per level 565 m

Cost/metre (Owner's cost) x $ 950/m (e)

Annual production rate x 145,950 tonnes/year

Tonnes accessed/level ÷ 445,500 tonnes/level

Total Ongoing Development Costs $175,844 (f)

3.22 Exploration Development

Enter 20% of Ongoing Capital Development above 35,169

3.23 Exploration Diamond Drilling

The annual cost of exploration diamond drilling is approximated by the following:

Quantity of drilling/setup:

No. of holes 4 x length/hole 60 = 240 (a)

Annual quantity of exploration development:

Total O.C.D. 175,844 (f) x 20% ÷ Cost/m 950 (e) = 37 (b)

Cost/metre drilled = 45.00 (c)

Setup interval = 25 (d)

Total Exploration Diamond Drilling Cost

$$= \frac{(a) \times (b) \times (c)}{(d)} = \$15,984$$

Reference Section No.	Item	Capital Cost $/Year
3.24	Equipment Replacement	

Total value of capital equipment $1,150,000
(From Form 3(a), Section 3.18)

Enter percentages from the graph and multiply by the total values above to estimate annual replacement costs by year.

	% of Total Value	Total Annual Replacement Costs
Year 1	2.5	28,750
Year 2	6.5	74,750
Year 3	10	115,000
Year 4	15	172,500
Year 5	18	207,000

FORM 4 EXAMPLE

REGIONAL COST FACTORS

Project Location Quebec - Use Blanket Factors

CAPITAL COST FACTOR

Cost Classification Area	Regional Cost Factor for ______ (From Table 4.3)		Percentage Distribution Capital Costs (Determined by User)		Multiplication Product	
Labour	______	x	______%	=	______	(a)
Plant & Equipment	______	x	______%	=	______	(b)
Materials & Consumables	______	x	______%	=	______	(c)
Hydro Power	______	x	______%	=	______	(d)
Transportation	______	x	______%	=	______	(e)
Provincial Tax	______	x	______%	=	______	(f)
		TOTAL =	100 %			

CAPITAL COST REGIONAL COST FACTOR (sum a to f) = ______

OPERATING COST FACTOR

Cost Classification Area	Regional Cost Factor for ______ (From Table 4.3)		Percentage Distribution Operating Costs (Determined by User)		Multiplication Product	
Labour	______	x	______%	=	______	(g)
Plant & Equipment	______	x	______%	=	______	(h)
Materials & Consumables	______	x	______%	=	______	(i)
Hydro Power	______	x	______%	=	______	(j)
Transportation	______	x	______%	=	______	(k)
Provincial Tax	______	x	______%	=	______	(l)
		TOTAL =	100 %			

OPERATING COST REGIONAL COST FACTOR (sum g to l) = ______

NOTE:

Forms 5(a) to 5(d) have not been included in this example since the geological and in-situ tonnes and grades were assumed (see Form 1). This information is entered on Form 5(e) in order to complete the Mineral Deposit Value calculation, Form 5(f).

FORM 5(e)

MINEABLE TONNES AND GRADE TO MILL

ADJUSTMENT FOR MINING RECOVERY

			Total In-Situ Reserve		Mining Recovery Factor		Actually Mined	
Tonnes		=	850,000	x	0.80	=	680,000	(a)
Primary Metal	- Units	=	8,500,000	x	0.80	=	6,800,000	(b)
	- Grade	=	10 g/t				10 g/t	(c)
Secondary Metal	- Units	=		x		=	N/A	(d)
	- Grade	=						(e)
Tertiary Metal	- Units	=		x		=	N/A	(f)
	- Grade	=						(g)

ADJUSTMENT FOR DILUTION

Dilution Factor = 1.20 (h)

Diluted Tonnes (a x h) = 816,000 (i)

"MINEABLE GRADES"

Primary Metal (c/h) = 8.33 (j)

Secondary Metal (e/h) = N/A (k)

Tertiary Metal (g/h) = N/A (l)

ADJUSTMENT FOR STOPE LOSSES

Stope Losses Factor = 0.90 (m)

"MINEABLE TONNES" to Mill (i x m) = 734,400 (n)

Units of Metal/Tonne in Mill Feed:

Primary Metal (b/i) = 8.33

Secondary Metal (d/i) =

Tertiary Metal (f/i) =

FORM 3(f)

MINERAL DEPOSIT VALUE

	Primary Metal	Secondary Metal	Tertiary Metal
Units of Metal/Tonne in Mill Feed	8.33		
Mill Recovery Factor	0.95		
Units of Metal Recovered Per Tonne of Mill Feed	7.91		
Current Metal Price/Unit	$15.11/g ($470 Cdn/oz)		
Value Per Tonne of Ore after Milling	$119.52		

TOTAL VALUE PER TONNE OF ORE AFTER MILLING = $119.52

NET VALUE PER TONNE OF ORE AFTER SMELTING/REFINING (Approximation only) = $112.00

FORM 6

PRELIMINARY CASH FLOW SUMMARY

Reference Section No.	Cash Flow (Initial 5 years)	$ in 1,000's					
	Year	0	1	2	3	4	5
Form 3(a)	Preproduction Capital	7,724					
Form 3(b)	Ongoing Capital		269	317	359	419	290
	Total Capital	7,724	269	317	359	419	290
Form 3(a) Section 3.16.1	Annual Production 145,950 tonnes						
Form 2(a)	Operating Cost/Tonne $64.34/tonne						
	Annual Operating Costs		9,390	9,390	9,390	9,390	9,390
	Total On-Site Costs	7,724	9,659	9,707	9,749	9,809	9,680
Form 2(a) Section 2.14	Freight Charges $ 2.00/tonne of Ore						
	Annual Freight Costs		292	292	292	292	292
	Total Costs	7,724	9,951	9,999	10,041	10,101	9,972
Form 5(f)	Revenue/Tonne after Smelting $112.00/tonne						
	Total Revenue		16,346	16,346	16,346	16,346	16,346
	* Cash Flow - By Year	-7,724	6,395	6,347	6,305	6,245	6,374
	* Cumulative Cash Flow	-7,724	-1,329	5,018	11,323	17,568	23,942

* Before depreciation and taxes.

APPENDIX B

APPENDIX B

BLANK CALCULATION FORMS

The following blank calculation forms are included in this section.

FORM 1 BASIC INFORMATION

FORM 2 OPERATING COSTS

- (a) OPERATING COSTS
- (b) MANPOWER SCHEDULE

FORM 3 CAPITAL COSTS

- (a) PREPRODUCTION CAPITAL COSTS
- (b) ONGOING CAPITAL COSTS

FORM 4 REGIONAL COST FACTORS

FORM 5 MINERAL DEPOSIT VALUE

- (a) SUMMARY OF GEOLOGICAL TONNES & GRADE
- (b) GEOLOGICAL TONNES & GRADE - RESERVE BY CROSS-SECTION
- (c) SUMMARY OF IN-SITU TONNES & GRADE
- (d) IN-SITU TONNES & GRADE - RESERVE BY CROSS-SECTION
- (e) MINEABLE TONNES & GRADE TO MILL
- (f) MINERAL DEPOSIT VALUE

FORM 6 PRELIMINARY CASH FLOW SUMMARY

FORM 1

BASIC INFORMATION

Estimate Prepared By: ____________________ Date: ______________

Name of Property: ____

Property Location: ____

Brief Description of Site and Local Area: ______________________________

Expected Overburden Conditions (Depth and Type): ______________________

Expected Rock Conditions: ______________________________________

Expected Ground Water Conditions: _______________________________

Other Relevant Information: ____________________________________

Attach additional sheets as required.

FORM 2(a)

OPERATING COSTS

SUMMARY (costs are developed on the next two pages)

	$/Tonne	
Stoping Costs	____	
Hoisting or Ramp Haulage Cost	____	
Level Haulage Cost	____	
General Mine Expense	____	
Surface Plant and Mine Services	____	
Staff and Administration	____	
Milling	____	
Subtotal	____	
Add Cost Contingency @ ___%	____	
Subtotal	____	(a)
Regional Operating Cost Factor	____	(b)
TOTAL OPERATING COST (a) x (b)	____	
Transportation of Mine Product	____	

FORM 2(a)

OPERATING COSTS

DETAILED CALCULATION FORM

Reference Section No.	Item	Operating Cost $/Tonne Ore
2.2	Selection of Production Rate	
	Rate selected ____________ tonnes/day	
	Mining days per year ______	
	Mining shifts per day _____	
2.3	Selection of Mining Method	
	Method selected ________	
2.4	Stoping Costs	______
2.5	Selection of Mine Access and Haulage Method	
	Access selected ________	
	Level haulage selected ________	
	Depth ________	
2.6	Hoisting or Ramp Haulage Cost	______
2.7	Level Haulage Cost	
	Haulage Distance ________	
	Haulage Capacity/Trip ________	
	Cost	______
2.8	General Mine Expense	______

2.9 Surface Plant and Mine Services

Power source selected ______

Cost

i) Labour ______
ii) Materials and Operating Costs ______
iii) Power ______
iv) Camp ______
v) Road Maintenance ______

Total Surface Plant and Mine Services ______

2.10 Staff and Administration ______

2.11 Milling

Selection of Location ______

Cost ______

2.14 Transportation of Mine Product

i) Bullion N/A

ii) Ore (enter cost from graph) ______

iii) Concentrate ______ \$/tonne (a)
(enter cost from graph)

Concentrating Ratio ______ (b)

Cost/tonne mined (a) ÷ (b) ______

3.20 Cost Contingency

3.20.1 Contingency for items omitted ______%
3.20.2 Contingency for variations in conditions ______%
3.20.3 Contingency for delays due to location ______%

Total Contingency Percentage, (add 3 lines above) ______%

Transfer all subsection totals to the summary page.

FORM 2(b)

MANPOWER SCHEDULE

Reference Section No.	Item	Manpower
3.21	Ongoing Capital Development	______
2.4	Stoping	
	Mining method selected ______	
	Productivity (tonnes/manshift) ______ (a)	
	Production rate (tonnes/day) ______ (b)	
	Manpower required (b)/(a)	______
2.6	Hoisting or Ramp Haulage	
	Hoisting:	
	Shifts worked per day ______ (c)	
	Manpower required per shift 2 (d)	
	Manpower required per day (c) x (d)	______
	Ramp:	
	Vertical depth ______ (e)	
	Productivity (tonnes/manshift) ______ (f)	
	Production rate (tonnes/day) ______ (g)	
	Manpower required (g)/(f)	______
2.7	Level Haulage	
	Haulage method selected ______	
	Manpower required	______
2.8	General Mine Expense	
	Track or trackless mine ______	______
	Subtotal Underground Manpower (including hoistman)	______
2.9	Surface Plant and Mine Services	______
2.10	Staff and Administration	______
2.11	Milling	______
	Subtotal Surface Manpower	______
	TOTAL ON-SITE MANPOWER	______

FORM 3(a)

PREPRODUCTION CAPITAL COSTS

SUMMARY (costs are developed on the next 8 pages)	$	
Feasibility Studies and Detailed Engineering		
Additional Diamond Drilling and Sampling		
Permits and Environmental Studies		
Project Management and Preproduction Scheduling		
Access to Minesite		
Site Preparation		
Camp Installation		
Site Services		
Electrical Power & Compressed Air		
Offices, Shops, Dry, Warehouse		
Mine Access		
Ancillary Shaft Excavations & Installations		
Hoisting Systems, Headframes & Bins		
Ventilation & Mine Air Heating		
Underground Development		
Underground Installations		
Equipment		
Concentrator		
Subtotal		
Add Cost Contingency @ _____%		
Subtotal		(a)
Regional Capital Cost Factor		(b)
TOTAL PREPRODUCTION CAPITAL COST (a) x (b)		

PREPRODUCTION CAPITAL COSTS - Detailed Calculation Form

Reference Section No.	Item	Capital Cost $
3.2	<u>Feasibility Studies & Detailed Engineering</u>	
	For shaft or ramp access? ________	
	Subsection Total (enter cost from graph)	______
3.3	<u>Additional Diamond Drilling & Sampling</u>	
3.3.1	<u>Drilling from surface</u>:	
	Number of holes ________	
	Average hole length ________ m	
	Cost/metre $________/m	
	Subtotal a) $________	
3.3.2	<u>Underground drilling</u>:	
	Number of holes ________	
	Average hole length ________ m	
	Cost/metre $________/m	
	Subtotal b) $________	
3.3.3	<u>Assaying samples</u>:	
	Number of samples ________	
	Cost per assay $________ ea.	
	Subtotal c) $________	
	Subsection Total (add a + b + c)	______

Reference Section No.	Item		Capital Cost $
3.4	Permits and Environmental Studies		
	Environmental sensitivity of region	________	
	Are harmful contaminants produced?	________	
	Do contaminants require 'normal' or 'special' handling?	________	
	Subsection Total (enter cost from graph)		______
3.5	Project Management and Preproduction Scheduling		
	Average monthly cost	$______/month	
	Duration of preproduction work	______ months	
	Subsection Total (multiply two lines above)		______
3.6	Access to Minesite		
3.6.1	New road construction: ______ km x $______/km =	$______	
3.6.2	Upgrading existing roads: ______ km x $______/km =	$______	
3.6.3	Road bridges: (total cost)	$______	
3.6.4/7	Other Access Costs:	$______	
	Subsection Total (add 4 lines above)		______
3.7	Site Preparation		

Site Category	Site Area m²	Cost $	
A	______	______	
B	______	______	
C	______	______	
D	______	______	
Total	______	______	______

Reference Section No.	Item	Capital Cost $
3.8	Camp Installation	
	Total manpower ____________ (see Form 2(b))	
	Camp capacity ____________ personnel	
	Subsection Total (enter cost from graph)	______
3.9	Site Services	
	Ramp or shaft access ________	
	Production rate ________ t.p.d.	
	Subsection Total (enter cost from graph)	______
3.10	Electrical Power & Compressed Air	
3.10.1	Electrical Power	
	Site power requirements ________ kW	
	Powerline:	
	Line Cost ___ km x $_____/km = $ ______	
	Site Cost $ ______	
	Total Powerline Cost $ ______	
	Generators: (enter cost from graph) = $ ______	
3.10.2	Compressor Plant	
	Compressed air requirements _____ m^3/sec.	
	Compressor inst'n. (enter cost from graph) $ ______	
	Subsection Total (add 3 lines above)	______
3.11	Offices, Shops, Dry, Warehouse	
	Subsection Total (enter cost from graph)	______

Reference Section No.	Item		Capital Cost $
3.12	Mine Access		
	Complete 3.12.1, 3.12.2, or 3.12.3 below.		
3.12.1	Shaft		
	Shaft type ____________		
	Shaft depth ________ m		
	Mobilize, setup, teardown, demobilize	$________	
	Shaft collar	$________	
	Shaft _____ m x $______/m =	$________	
	Shaft changeover to skipping =	$________	
	Total Shaft Costs (add 4 lines above)		________
3.12.2	Decline		
	Depth of lowest level _______ m		
	Mobilize, setup, teardown, demobilize	$______	
	Decline portal	$______	
	Decline excavation	$______	
	Total Decline Costs (add 3 lines above)		________
3.12.3	Adit(s)		
	Mobilize, setup, teardown, demobilize	$______	
	Adit portal	$______	
	Adit excavation ____ m x $_____/m =	$______	
	Internal ramp (+15%) ____ m x $_____/m =	$______	
	Internal ramp (-15%) ____ m x $_____/m =	$______	
	Total Adit Costs (add 5 lines above)		________

Reference Section No.	Item		Capital Cost $
3.13	Ancillary Shaft Excavations & Installations		
3.13.1	Shaft stations: _____ x $_____ ea. =	$_____	
3.13.2	Loading pocket: =	$_____	
3.13.3	Lip pockets: _____ x $_____ ea. =	$_____	
3.13.4	Spill handling: =	$_____	
3.13.5	Shaft bottom construction: =	$_____	
	Subsection Total (add 5 lines above)		$_____
3.14	Hoisting System, Headframe & Bin		
	Hoisting depth _____ m		
	Hoisting capacity (ore & waste) _____ tonnes/hour		
3.14.1	Hoist & Hoistroom		
	Hoist selected:		
	Motor size _____ kw		
	Drum diameter _____ ft.		
	Total cost, hoist & hoistroom	$_____	
3.14.2	Headframe & Collarhouse		
	Headframe height _____ m		
	Total cost, headframe & collarhouse	$_____	
3.14.3	Headframe Bins		
	Bin or 'dump area' _____		
	If bin, what size? _____ tonnes		
	Total cost, bin or dump area	$_____	
3.14.4	Conveyances		
	Conveyance combination _____		
	Total cost, conveyances	$_____	
	Subsection Total (add 4 lines above)		$_____

Reference Section No.	Item	Capital Cost $
3.15	<u>Ventilation & Mine Air Heating</u>	
3.15.1	Primary ventilation fans $ ______	
3.15.2	Mine air heaters $ ______	
	Subsection Total (add 2 lines above)	$ ______
3.16	<u>Underground Development</u>	
3.16.1	<u>Level Development</u>	
	Production level development costs are approximated by the following:	
	<u>Quantity of development per level</u>:	
	Avg. x-cut length ___ m + strike length ____ m = ____ (a)	
	Cost per metre ____ (b)	
	<u>Annual Production Tonnage</u>:	
	Multiply the two items below:	
	Daily production rate _____ t.p.d.	
	Working days per year _____ days	
	Annual production tonnage = ____ (c)	
	<u>Tonnes accessed per level</u>:	
	Multiply the four items below:	
	Strike length between mining limits _____ m	
	Average stoping width _____ m	
	Ore length between main levels _____ m	
	Ore tonnage factor _____ t/m^3	
	Tonnes accessed = ____ (d)	
	Total Preproduction Level Development Cost $= \frac{(a) \times (b) \times (c) \times 2}{(d)}$ $ ______	

Reference Section No.	Item	Capital Cost $
3.16.2	<u>Ore Pass System</u>	
	Distance between top & bottom production levels ____ m	
	Total cost, (enter cost from graph) $______	
3.16.3	<u>Primary Ventilation & Escapeway</u>	
	Depth of bottom level below surface _____ m	
	Total cost, (enter cost from graph) $______	
	Subsection Total (add 3 totals above)	________
3.17	<u>Underground Installations</u>	
3.17.1	Main Sumps and Pump Stations:	
	Is mine 'dry', 'average' or 'wet'? ________	$______
3.17.2	Rockbreaker & Grizzly:	$______
3.17.3	Ore Pass Controls:	
	No. of controls _____ x $______/control	$______
3.17.4	Underground Electrical Room/Load Centre:	
	No. of installations _____ x $______/inst'n.	$______
3.17.5	Miscellaneous Installations:	
	No. of levels _____ x $______/level	$______
	Subsection Total (add 5 lines above)	________
3.18	<u>Equipment</u>	
	Shaft, ramp or adit? ______	
	Track or trackless haulage? ______	
	Subsection Total (enter cost from graph)	$________

Reference Section No.	Item		Capital Cost $
3.19	Concentrator		
3.19.1	Concentrator construction: Process type ______		
3.19.2	Construction costs, (enter cost from graph)	$ ______	
	Tailings disposal area: Construction costs, (enter cost from graph)	$ ______	
	Subsection Total, (add 2 lines above)		______
3.20	Cost Contingency		
3.20.1	Contingency for items omitted	______ %	
3.20.2	Contingency for variations in conditions	______ %	
3.20.3	Contingency for delays due to location	______ %	
	Total Contingency Percentage, (add 3 lines above)		______ %

Transfer all subsection totals to the summary page.

FORM 3(b)

ONGOING CAPITAL COSTS

SUMMARY (costs are developed on the next two pages)

	Year 1	Year 2	Year 3	Year 4	Year 5
Ongoing Capital Development					
Exploration Development					
Exploration Diamond Drilling					
Equipment Replacement					
Subtotal					
Regional Cost Factor					
TOTAL ONGOING CAPITAL COST					

ONGOING CAPITAL COSTS - Detailed Calculation Form

Reference Section No.	Item	Capital Cost $/Year

3.21 Ongoing Capital Development (O.C.D.)

Calculation is similar to that used for preproduction level development, Form 3(a), Section 3.16.1, except that results are NOT multiplied by 2.

Ongoing Capital Development Costs can be approximated by the following:

Quantity of development per level _______ m
Cost/metre (Owner's cost) x $_______/m (e)
Annual production rate x _______ tonnes/year
Tonnes accessed/level ÷ _______ tonnes/level
Total Ongoing Development Costs $______ (f)

3.22 Exploration Development

Enter 20% of Ongoing Capital Development above _______

3.23 Exploration Diamond Drilling

The annual cost of exploration diamond drilling is approximated by the following:

Quantity of drilling/setup:
No. of holes _____ x length/hole _____ = ______ (a)

Annual quantity of exploration development:
Total O.C.D. _____ (f) x 20% ÷ Cost/m ____ (e) = ______ (b)

Cost/metre drilled = ______ (c)

Setup interval = ______ (d)

Total Exploration Diamond Drilling Cost

$$= \frac{(a) \times (b) \times (c)}{(d)} = \$\_\_\_\_\_\_$$

Reference Section No.	Item	Capital Cost $/Year
3.24	Equipment Replacement	

Total value of capital equipment $ ______
(From Form 3(a), Section 3.18)

Enter percentages from the graph and multiply by the total values above to estimate annual replacement costs by year.

	% of Total Value	Total Annual Replacement Costs
Year 1	______	______
Year 2	______	______
Year 3	______	______
Year 4	______	______
Year 5	______	______

FORM 4

REGIONAL COST FACTORS

Project Location ______________

CAPITAL COST FACTOR

Cost Classification Area	Regional Cost Factor for ____ (From Table 4.3)		Percentage Distribution of Capital Costs (Determined by User)		Multiplication Product	
Labour	______	x	______%	=	________	(a)
Plant & Equipment	______	x	______%	=	________	(b)
Materials & Consumables	______	x	______%	=	________	(c)
Hydro Power	______	x	______%	=	________	(d)
Transportation	______	x	______%	=	________	(e)
Provincial Tax	______	x	______%	=	________	(f)
		TOTAL =	100 %			

CAPITAL COST REGIONAL COST FACTOR (sum a to f) = ________

OPERATING COST FACTOR

Cost Classification Area	Regional Cost Factor for ____ (From Table 4.3)		Percentage Distribution of Operating Costs (Determined by User)		Multiplication Product	
Labour	______	x	______%	=	________	(g)
Plant & Equipment	______	x	______%	=	________	(h)
Materials & Consumables	______	x	______%	=	________	(i)
Hydro Power	______	x	______%	=	________	(j)
Transportation	______	x	______%	=	________	(k)
Provincial Tax	______	x	______%	=	________	(l)
		TOTAL =	100 %			

OPERATING COST REGIONAL COST FACTOR (sum g to l) = ________

FORM 5(a)

GEOLOGICAL TONNES & GRADE

SUMMARY - From Forms 5(b)		Unit of Measurement
Primary metal is	____________	____________
Secondary metal is	____________	____________
Tertiary metal is	____________	____________

Cross Section No.	Total Tonnes of Mineralization	Total Units of Metal		
		Primary Metal	Secondary Metal	Tertiary Metal
	(A)	(B)	(C)	(D)
TOTALS				

TOTAL GEOLOGICAL RESERVE = ____________ (A)

GEOLOGICAL RESERVE GRADE:

Primary Metal (B/A) = ____________

Secondary Metal (C/A) = ____________

Tertiary Metal (D/A) = ____________

FORM 5(b)

GEOLOGICAL TONNES & GRADE

MINERAL RESERVE BY CROSS-SECTION

Cross-Section No. ____________

	Mineralized Area			
	A	B	C	D
Diamond Drill Hole No.				
Area of Mineralization (m²)				
Grade of Mineralized Area:				
Primary Metal				
Secondary Metal				
Tertiary Metal				
Strike Length Represented by Cross-section (m)				
Volume of Mineralization (m³)				
Tonnage Factor (tonnes/m³)				
Tonnes of Mineralization				
Total Units of Metal:				
Primary Metal				
Secondary Metal				
Tertiary Metal				

TOTAL TONNES (A + B + C + D) = ______________

	Total Units of Metal	Average Grade
Primary Metal		
Secondary Metal		
Tertiary Metal		

FORM 5(c)

IN-SITU TONNES & GRADE

SUMMARY - From Forms 5(d)	Unit of Measurement
Primary metal is ____________	____________
Secondary metal is ____________	____________
Tertiary metal is ____________	____________

Cross Section No.	Total In-Situ Tonnes	Total Units of Metal		
		Primary Metal	Secondary Metal	Tertiary Metal
	(A)	(B)	(C)	(D)
TOTALS				

TOTAL IN-SITU RESERVE = ____________ (A)

IN-SITU RESERVE GRADE:

Primary Metal (B/A) = ____________

Secondary Metal (C/A) = ____________

Tertiary Metal (D/A) = ____________

FORM 5(d)

IN-SITU TONNES & GRADE

IN-SITU RESERVE BY CROSS-SECTION

Cross-Section No. ____________

	Mining Zone			
	A	B	C	D
AREAS (Within Mining Limits)				
Area of Mineralization (m^2)				
Area of Waste (m^2)				
Total Area (m^2)				
VOLUMES (Within Mining Limits)				
Strike Length Represented by Cross-section (m)				
Volume of Mineralization (m^3)				
Volume of Waste (m^3)				
Total Volume (m^3)				
TONNES (Within Mining Limits)				
Tonnage Factor - Ore (tonnes/m^3)				
Tonnage Factor - Waste (tonnes/m^3)				
Tonnes of Mineralization				
Tonnes of Waste				
Tonnes of Mineralization & Waste (Ore)				

TOTAL TONNES OF ORE - ALL ZONES ____________ (a)

GRADES OF MINERALIZED AREAS (Geological Grade)				
Primary Metal				
Secondary Metal				
Tertiary Metal				
UNITS OF METAL (Tonnes of Mineralization x Geological Grade)				
Primary Metal				
Secondary Metal				
Tertiary Metal				

	Total Units All Zones (b)	Average Grade All Zones (b) ÷ (a) (within mining limits)
Primary		
Secondary		
Tertiary		

FORM 5(e)

MINEABLE TONNES AND GRADE TO MILL

ADJUSTMENT FOR MINING RECOVERY

			Total In-Situ Reserve		Mining Recovery Factor		Actually Mined	
Tonnes		=	________	x	______	=	________	(a)
Primary Metal	- Units	=	________	x	______	=	________	(b)
	- Grade	=	________				________	(c)
Secondary Metal	- Units	=	________	x	______	=	________	(d)
	- Grade	=	________				________	(e)
Tertiary Metal	- Units	=	________	x	______	=	________	(f)
	- Grade	=	________				________	(g)

ADJUSTMENT FOR DILUTION

Dilution Factor = ________ (h)

Diluted Tonnes (a x h) = ________ (i)

"MINEABLE GRADES"

Primary Metal (c/h) = ________ (j)
Secondary Metal (e/h) = ________ (k)
Tertiary Metal (g/h) = ________ (l)

ADJUSTMENT FOR STOPE LOSSES

Stope Losses Factor = ________ (m)

"MINEABLE TONNES" to Mill (i x m) = ________ (n)

Units of Metal/Tonne in Mill Feed:

Primary Metal (b/i) = ________
Secondary Metal (d/i) = ________
Tertiary Metal (f/i) = ________

FORM 5(f)

MINERAL DEPOSIT VALUE

	Primary Metal	Secondary Metal	Tertiary Metal
Units of Metal/Tonne in Mill Feed			
Mill Recovery Factor			
Units of Metal Recovered Per Tonne of Mill Feed			
Current Metal Price/Unit			
Value Per Tonne of Ore after Milling			

TOTAL VALUE PER TONNE OF ORE AFTER MILLING = ____________

NET VALUE PER TONNE OF ORE AFTER SMELTING/REFINING (Approximation only) = ____________

FORM 6

PRELIMINARY CASH FLOW SUMMARY

Reference Section No.	Cash Flow (Initial 5 years)	$ in 1,000's					
	Year	0	1	2	3	4	5
Form 3(a)	Preproduction Capital	____					
Form 3(b)	Ongoing Capital		____	____	____	____	____
	Total Capital	____	____	____	____	____	____
Form 3(a) Section 3.16.1	Annual Production _____ tonnes						
Form 2(a)	Operating Cost/Tonne $______/tonne						
	Annual Operating Costs		____	____	____	____	____
	Total On-Site Costs	____	____	____	____	____	____
Form 2(a) Section 2.14	Freight Charges $______/tonne of Ore						
	Annual Freight Costs		____	____	____	____	____
	Total Costs	____	____	____	____	____	____
Form 5(f)	Revenue/Tonne after Smelting $_____/tonne						
	Total Revenue		____	____	____	____	____
	* Cash Flow - By Year	____	____	____	____	____	____
	* Cumulative Cash Flow	____	____	____	____	____	____

* Before depreciation and taxes.