GEOLOGY AND MINE PLANNING

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1. INTRODUCTION

This dissertation aims to review the main aspects involved in mine planning, in order to provide the geologists with the main criteria to obtain a preliminary estimate of the minability of an in situ orebody, and to make the geologists aware of the information they can contribute to the planning engineers. Furthermore, an understanding of the basic principles behind mine planning may help the exploration geologist to select more realistic targets of exploration.

The subject is extensive and, unfortunately, the attempts to generalize and summarize very often result in oversimplification of some of the aspects involved. For the same reason, this review has been limited mostly to metalliferous mining.

The geological aspects discussed in the text can be grouped in two main categories: those geological factors limiting the applicability of the different methods and affecting the mine layout planning, and those geological aspects affected by the choosing of a particular mining method (e.g. ore reserve delineation and estimation, minability, grade cutoffs, ore recovery, dilution, grade control, etc.).

The most obvious reasons why the accurate delineation of the ore-bodies, and the determination of their geological and rock-mechanical characteristics are needed, are that they constitute basic inputs in the planning of the mine and in the estimation of its economic feasibility. However, the tendency of recent years to increase productivity through mechanization of the mining operations has, in a particular way, accentuated the need for increased accuracy in the data-gathering work of the geologists.

Modern high productivity mining methods are capital-cost-intensive: they require high investments of capital in equipment as well as in development, infrastructure, etc.. In the past, the methods were labour and, sometimes, supplies (explosives, timber) intensive; under these circumstances, the overall grade and tonnage produced in the mine could be controlled by stopping or starting the mining of certain stopes of given grades, or by playing with their mining rates: it was just a matter of moving the labour to other stopes.

Nowadays, besides the rock mechanics problems created when leaving the larger-span (required by high-productivity equipment) stopes idle for some time, there is a larger investment in development and infrastructure that will not produce returns, and this is going to affect negatively the overall profitability of the operation. The profitability derived from the use of expensive equipment is based on a maximum use of that equipment (as close to 100% of the time as possible), and it also suffers if the equipment has to constantly move from one stope to another to achieve the required blend; modern high-productivity stoping requires that, in each stope, a maximum output (for a given assemblage of equipment) must be achieved and maintained. Even labour has changed with mechanization, becoming more specialized and hence less flexible to move to different tasks.

Therefore, the only tool left to produce an optimum output (of desired grades and tonnages) throughout the life of the mine is the careful scheduling of the mining of the stopes, whose grades and tonnages need to be accurately known (not only the overall grade and tonnage of each stope, but also - according to the detail of the mining schedule - the evolution of the recoverable grades and tonnages throughout the productive life of each stope). Contingency mining plans should be considered in an overall schedule, so as to mine at a higher cutoff grade (always at the maximized stope outputs) in times of higher product prices (see sections 2.1 and 2.4).

2. MINE PLANNING - SOME PRELIMINARY CONSIDERATIONS

2.1. Cutoff grades

One of the first problems faced when trying to delineate mineable orebodies is that of defining the "cutoff grade". The following section is based on H.K. Taylor's (1972) paper on the general background theory of cutoff grades.

The "cutoff grade" is an operating control and it should not be understood as a "pay limit" or "breakeven grade", which are the grade equivalents of particular financial specifications. The cutoff grade is any grade that, for any specified reason, is used to separate two courses of action, eg. to mine or to leave, to mill or to dump. The breakeven grade is that grade from which the recoverable revenue exactly balances the costs of mining, treatment and marketing, however these may be sub-defined.

In the past, the average-cost breakeven grades have been convention-ally used as cutoff grades. This is best exemplified by the South African government-enforced "pay limit" system (Taylor, 1978; Storrar, 1977), in which the "pay limit" is defined as that grade of an ore at which it can be mined and treated without profit or loss, i.e. when the revenues obtained balance the mining and treatment working costs, "and, possibly, also the costs of transporting the marketable product to the delivery point". The pay limit is used thereafter to classify all superior grades as "payable" and those inferior as "unpayable". The South African pay limit system requires (i) that all ore mined must be able to bear its proportionate share of working costs, including overhead and (sometimes) development expenses; (ii) that the average "breakeven" or "pay-limit" grade so determined be considered as the only legitimate cutoff grade, and any better ore automatically becomes part of the official ore reserve; (iii) that the overall stoping must be done almost exactly at the average grade of the whole ore reserve; and (iv) that in these matters, time and capital expenditures be considered wholly irrelevant. The pure break-even system remains in use because of its simplicity and fair reliability, but it is affected by some problems and restrictions.

This conventional system yields a single static criterion that is independent of quàntity, time or locality, and it may conflict with the need for several types of cutoff grades, eg. those for entire orebodies, for lateral fringes, for width increments, for waste diluted ore drawn in bulk mining systems, and for the separation of already mined open pit rock.   Problems arise with this conventional system when it comes to "sunk" costs, considered excludable from breakeven determination but which have still to be paid somehow, and in the separation of "fixed" and "variable" costs when the rate of ore output is changed.   Furthermore, the average breakeven grade fails completely as a cutoff if changing it alters the ore availability sufficiently to affect the rate of output, and, thus, the unit costs upon which the breakeven policy must necessarily be based.

If the product price raises, the breakeven grade decreases and the mine would profit from ore that would stay unmined at the lower price. However, a greater revenue - and almost always a greater profit - results from selling the maximum weight of product at the higher price, even at the permanent sacrifice of the lower-grade material.   The profit from mining the additional low-grade ore in a high-price period, is always less than the profit lost by deferring the mining of the same tonnage of higher-grade ore to a later lower-price period.   Higher prices call for more product to the market, that, in absence of spare milling capacity, must be met by raising cutoff and average grades.

A general cutoff grade theory should include both breakeven and balance-of-operation principles.

Taylor (1972) has developed a cutoff grade theory based on the division of the mining process into three main stages, each having distinctive variable costs and costing units.   The first stage consists in making the ore available for selection, by development in an underground mine or by mining in an open pit;   advance (or capital) costs per ton made available for selection are characteristic of this stage. The second stage is that of the handling and treatment of the selected ore, by stoping, haulage and hoisting, and by concentration;   its costs are ore-variable costs per ton of ore milled.   The third stage consists

in the handling treatment and marketing of the product, by smelting, transport, direct sale,etc.; its costs are product-variable costs per ton of product sold. Superimposed are fixed costs per unit of time, including both administration costs and any fixed-cost elements within the stages themselves.

The objective of a mine policy may be to maximize the total profit, the present value of the total profit, the profit per unit of time, or the discount rate itself (although, for the last objective, in respect of cutoff policy its requirements are similar to those for maximum present value). However, it is impossible to achieve true maximization of the present value with a cutoff grade that is constant over time.

If the total profit from a finite mineral inventory must be maximized, the profit per "most-seriously-limiting-factor" in each of the three stages of the mining process must be maximized. Thus,

For	Maximize
Limited ore availability	Profit per unit made available for selection
Limited ore handling or treatment capacity	Profit per unit of ore treated.
Limited product treatment or marketing capacity	Profit per unit of product sold.

Each condition is satisfied by the use of a particular cost breakeven grade. For the purpose of this breakeven analysis product-variable costs are best treated as deductions from revenue, fixed costs behave as if borne wholly by the limiting stage, and since the ore is depleted as surely by abandonment as by extraction and milling advance (or capital) costs (whether incurred already, or to be re-incurred in replacement) are irrelevant to choice. So are fixed costs if the ore availability is the effective limit.

Let Q_a , Q_o , Q_p be quantities available, treated and produced per unit of time; W_a , W_o , W_p be costs per unit of Q, respectively; W_f be fixed costs per unit of time; S_p be price per unit of Q_p (product); $Q_{a(max)}$, $Q_{o(max)}$, $Q_{p(max)}$ be limiting quantities per unit of time; and B_a , B_o , B_p be the respective breakeven grades of ore.

Then,

$$B_a = W_o/(S_p - W_p) \quad [= \text{ore-variable cost/net product price}]$$
$$B_o = (W_o + W_f/Q_{o(max.)})/(S_p - W_p) \quad [= \text{(ore-variable cost plus unit fixed cost)/net product price}]$$
$$B_p = W_o/(S_p - W_p - W_f/Q_{p(max.)}) \quad [= \text{ore-variable cost/ (net product price minus unit fixed cost)}]$$

For profitable conditions, $B_o > B_p > B_a$

For each pair of stages the optimum cutoff grade is the balancing grade defined by the two breakeven grades. Of the three balancing cutoff grades Z_{ao}, Z_{op}, Z_{ap}, the middle value gives the overall optimum cutoff grade (Z) that will maximize the total undiscounted profit of the mineral inventory.

To illustrate the theory, Taylor (op. cit.) considered the following example: an ore inventory corresponding to a lognormal population, with standard deviation 0.50 (in natural logarithms) plus 50% extraneous waste (or its equivalent in unproductive development work), a mean grade of 4.43%, a geometric mean of 3.97% and a resource availability of 1000 tons. The other constraints are as follows: availability rate, 100 ton/month; advance cost, $0.40 per ton; product price, $100 per ton; milling rate, 25 ton/month; ore-variable cost, $1.50 per ton; product-variable cost, $15 per ton; sales rate, 1.25 ton/month; fixed cost, $15 per month.

Lognormality is often an adequate approximation to the distribution of the grades in populations of selectable ore "parcels", within orebodies other than those physically most rich ("parcel" is a body of mineable or treatable material, large enough to be wholly selected or wholly rejected by the mining or selection system in use). A feature of the lognormal distribution is an almost perfectly linear relation between the average grade above cutoff (G) and the cutoff grade (Z), for all cutoffs above the geometric mean (θ). The calculated gradients (ΔG/ΔZ) and other ratios for different standard deviations are presented in the table in Fig. 2.1 and in Fig. 2.2. For a lognormal population, with a known mean (M) and standard deviation (σ), given a cutoff (Z) higher than the geometric mean (θ) it is possible to estimate the tonnage ratio above cutoff, and the average grade above cutoff (G), from the graph in Fig. 2.2.

FIG.2.1 (Taylor,1972)

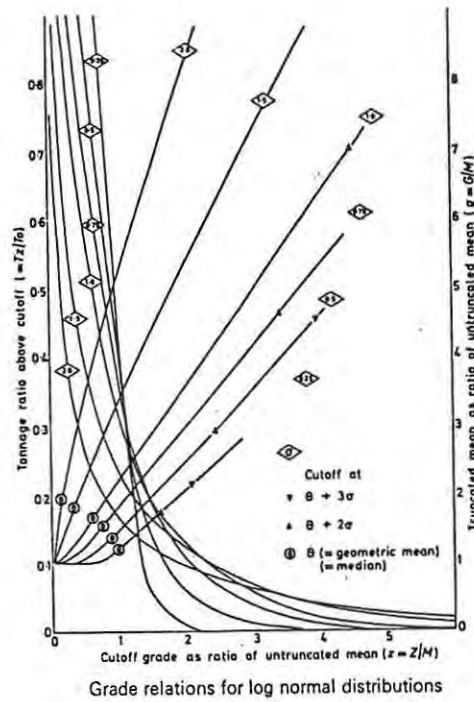| Standard deviation. σ	0·25	0·50	0·75	1·00	1·50	2·00
Variance, σ^2	0·06	0·25	0·56	1·00	2·25	4·00
Gradient. $\Delta G/\Delta z$	0·9	1·05	1·2	1·4	2·0	3·1
Population above θ	0·5	0·5	0·5	0·5	0·5	0·5
Content above θ	0·60	0·69	0·77	0·84	0·93	0·98

Grade relations for log normal distributions

FIG.2.2 (Taylor,1972)

Knowing the tonnage and average grade of the resource above different cutoff grades, and the costing relationships, we can calculate the profits of the three stages of the mining process for the whole range of cutoff grades. Thus, the availability rate (100 tons/month), the ore handling or milling rate capacity (25 tons/month), and the sales of product (1.25 tons/month) are successively fixed and the profits for the corresponding stages are calculated as a function of the cutoff grade. Plotting the results yields the three continuous-line graphs given in Fig. 2.3. Each line peaks at the stage breakeven grade (i.e. B_α, B_o and B_p), and each pair of lines crosses at the balancing cutoff grade (i.e. $Z_{\alpha p}$, $Z_{\alpha o}$ and Z_{op}). The graph for operation under the simultaneous three limits must clearly be that indicated by the thickened line (whose details are tabulated in Fig. 2.4), and its peak coincides with the optimum cutoff $Z = 2.55\%$.

All mines have several different selectable populations, with different ore characteristics (including recoveries and dilutions) and costing relationships, and they should be analyzed individually (although they are interdependent in some aspects that should be considered accordingly).
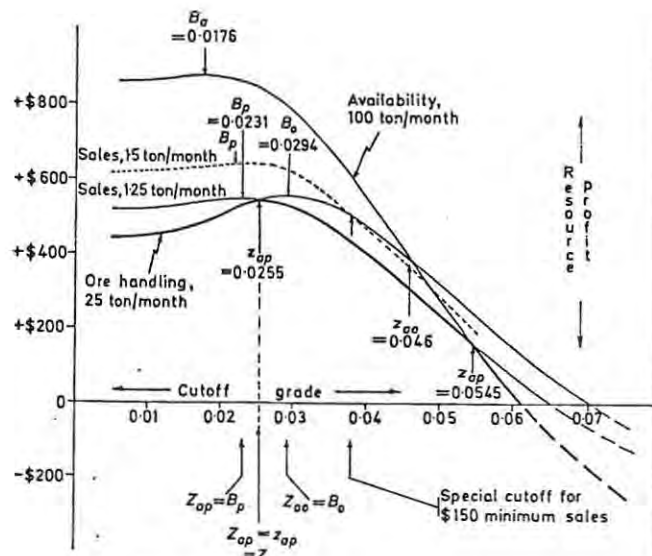
General case
FIG.2.3 (Taylor,1972)

A general case

Log normal + 50% waste
Mean = 0·0443, θ = 0·0397,
σ = 0·50, tons = 1000

Availability rate, 100 ton/month; advance cost, $0·40/ton; product price, $100/ton; milling rate, 25 ton/month; ore-variable cost, $1·50/ton; product-variable cost, $15/ton; sales rate, 1·25 ton/month; fixed cost, $25/month

| Cutoff grade	Average grade	Resource tons	Availability rate, ton/month	Advance cost, $/month	Milling rate, ton/month	Ore-variable cost, $/month	Fixed cost, $/month	Mine cost, $/month	Gross revenue, $/month	Pro-variable cost, $/month	Net revenue, $/month	Profit, $/month	Resource life, months	Resource profit, $	Average cost, $/ton*	Apparent breakeven
0·075	0·0974	59	100	40	5·9	8·9	25	73·9	57·5	8·6	48·9	(25·0)	10	(250)	12·52	0·147
0·070	0·0916	77	100	40	7·7	11·5	25	76·5	70·5	10·6	59·9	(16·6)	10	(166)	9·94	0·117
0·065	0·0866	97	100	40	9·7	14·6	25	79·6	84·0	12·6	71·4	(8·2)	10	(82)	8·20	0·096
0·060	0·0808	128	100	40	12·8	19·2	25	84·2	103·4	15·5	87·9	3·7	10	37	6·58	0·077
0·055	0·0760	161	100	40	16·1	24·1	25	89·1	122·4	18·4	104·0	14·9	10	149	5·54	0·065
0·050	0·0710	205	85·9	34·4	17·6	26·4	25	85·8	125·0	18·7	106·3	20·5	11·6	238	4·87	0·057
0·045	0·0660	260	72·8	29·2	18·9	28·4	25	82·6	125·0	18·7	106·3	23·7	13·7	325	4·37	0·051
0·040	0·0613	325	62·7	25·1	20·4	30·6	25	80·7	125·0	18·7	106·3	25·6	15·9	407	3·96	0·047
0·035	0·0569	400	54·9	22·0	22·0	33·0	25	80·0	125·0	18·7	106·3	26·3	18·2	479	3·64	0·043
0·030	0·0529	479	49·3	19·7	23·6	35·5	25	80·2	125·0	18·7	106·3	26·1	20·3	529	3·40	0·040
0·025	0·0496	550	45·5	18·2	25	37·5	25	80·7	124·0	18·6	105·4	24·7	22·0	544	3·23	0·038
0·020	0·0468	614	40·7	16·3	25	37·5	25	78·8	117·0	17·5	99·5	20·7	24·6	507	3·15	0·037
0·015	0·0451	651	38·4	15·3	25	37·5	25	77·8	112·7	16·9	95·8	18·0	26·0	468	3·11	0·036
0·010	0·0444	665	37·5	15·0	25	37·5	25	77·5	111·0	16·6	94·4	16·9	28·6	450	3·10	0·036
0·005	0·0443	666	37·5	15·0	25	37·5	25	77·5	110·7	16·6	94·1	16·6	26·7	447	3·10	0·036
0·000	—	1000														

*Excluding product-variable cost.

FIG.2.4 (Taylor,1972)

Further details and discussion on this cutoff grade theory (very superficially summarized here) can be found in Taylor's paper. The problem is complex and not completely solved, yet. Blackwell (1971) applied principles similar to those of Taylor to the determination of an optimum cutoff grade at the Bougainville porphyry copper deposit.

2.2. Choosing a mine-mill capacity

Norén's (1971) approach to the determination of the optimum rate
of production consists in maximizing the Net Present Value ("capital
value of the deposit", in the graph in Fig. 2.5), discounted back to
year 0 at a fixed rate of interest, for the different alternative rates
of production (see Fig. 2.5).   Then, successively changing the rate of
interest, new optimum rates of production are similarly obtained;   when
plotted against the rates of interest, we obtain a graph similar to the
one in Fig. 2.6, which permits to select the optimum rate of production
for any rate of interest.
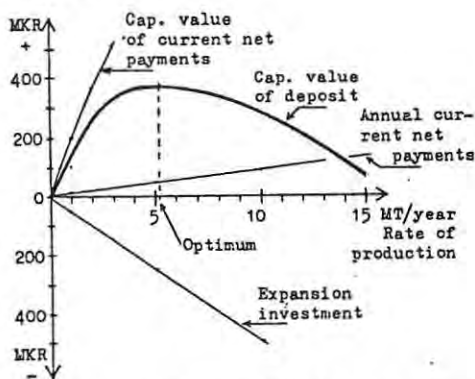
Capital values at time 0 of the ore deposit
at different annual rates of production (compare Table 1).

FIG.2.5   (Norén,1971)
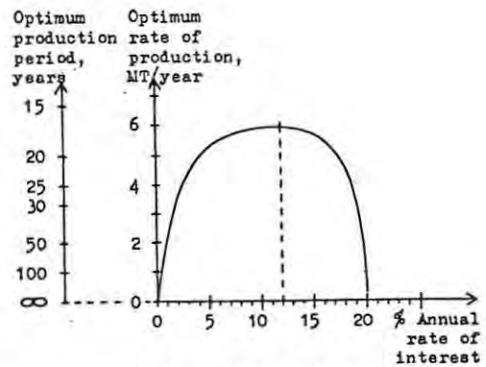
Optimum rates of production and optimum
production periods at different rates of interest. (Other
data according to the example of Table 1.)

FIG.2.6   (Norén,1971)

A characteristic of Norén's program is that the optimization of the
rate of production always produces a schedule that leads to the total
mining of the ore reserves, and these optimal mining rate schedules can
generate negative cash flows during the last years of mining only to
meet the model constraints (even when debt repayment requirements have
been fulfilled).   The goal set should not be mining a fixed amount of
rock at a maximum benefit, but really to obtain maximum benefit from
development of the deposit, mined out or not.   Lizotte and Elbrond
(1982) presented a method to choose an optimum mine-mill capacity and
production schedules using open-ended dynamic programming.   A dynamic
approach is necessary since even the economic goals pursued vary with
time: at the planning stage decisions can be taken so as to maximize
the total discounted cash flows, whereas the management of a producing
mine may seek to minimize costs over a given period of time under

different constraints than originally planned for.   Testing with
particular cyclical price variations and cost structures led to an
open-ended dynamic programming.   The model allows the optimization of
two decision variables: The sequence of annual mining rates and the
cutoff grades associated with the mining rates.

The four evaluation criteria, used by Lizotte and Elbrond to compare
the economic outcomes obtained by computer program runs, are the Net
Present Value (NPV), the Present Value Ratio (PVR; i.e. the ratio of the
present value of the cash inflows to the present value of the investments),
the Internal Rate of Return (IRR) or Discounted Cash Flow Rate of Return
(DCFROR), and the Wealth Growth Rate (WGR).

The Wealth Growth Rate (WGR) is the interest rate that equals the
future value of capital outlays to the future value of generated cash
flows, reinvested at a stipulated interest rate i.   This criterion
considers the time value of cash flows, but the present values are not
computed.   For a single initial investment Io, WGR is the interest rate,
g, that satisfies the following equation:

$$Io \, (1+g)^n = \sum_{x=1}^{n} CFx(1+i)^{n-x}$$

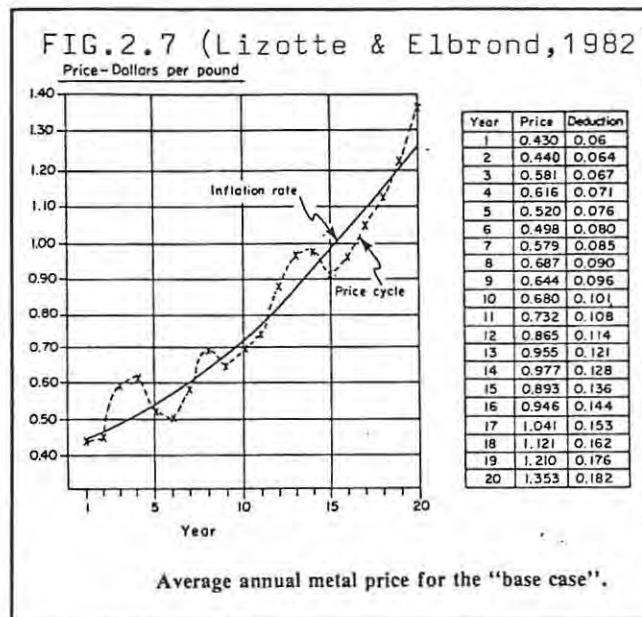When the interest payments are continuous over time, the WGR can
be computed as follows:

$$WGR = i + (Ln(PVR))/n$$

where PVR is the Present Value Ratio, n is the economic life of the
project, i is the interest rate used in computing the PVR, and Ln is the
natural logarithm.   This equation demonstrates that the WGR can be used
to compare projects having different economic lives and requiring different
investments.

WGR is somewhat of a logical amalgam of PVR and IRR that takes into
account the various economic lives of the projects compared.   WGR is
easy to calculate, by approximating with the formula above, and is the
main criterion for classifying mine-mill capacities by their economic
worth.   IRRs should be based on differential investments and cash flows
should have to be computed to be correctly used to compare the projects.

The NPV is not a valid classifier because it does not reflect the
different investments required by the different mine-mill capacity
alternatives, and PVR does not implicitly state the earning power and
the different economic lives of the projects evaluated.

The cash flows discounted were computed in nominal monies (monies
having various purchasing powers), using forecasts same as the one for
metal prices shown in Fig. 2.7, which accounts for inflation and price
cycles (the price cycles shown are similar to those of the zinc price
between 1953 and 1972, but adjusted to contemporary values and annually
inflated using an average 5% rate).

FIG.2.7 (Lizotte & Elbrond,1982)
Price-Dollars per pound

Year	Price	Deduction
I	0.430	0.06
2	0.440	0.064
3	0.581	0.067
4	0.616	0.071
5	0.520	0.076
6	0.498	0.080
7	0.579	0.085
8	0.687	0.090
9	0.644	0.096
10	0.680	0.101
11	0.732	0.108
12	0.865	0.114
13	0.955	0.121
14	0.977	0.128
15	0.893	0.136
16	0.946	0.144
17	1.041	0.153
18	1.121	0.162
19	1.210	0.176
20	1.353	0.182

Average annual metal price for the "base case".

2.3. High-grading

As a theory of cutoff grades is necessary to delineate the orebodies
and economic criteria are used to choose an optimum mine-mill capacity,
the principle of high-grading is generally applied in the planning of the
production schedule.   These three decision variables are interrelated
and must be considered in a general mine plan.

The conventionally accepted approach in selecting a production schedule for feasibility study purposes, is to plan the extraction at a constant average grade throughout the life of the mine. The average grade corresponds to the mean (for a given distribution of values in a population of "parcels") of the values above an economic cutoff grade.

However, it is generally accepted (a few exceptions are mentioned later) that high-grading can result in an increased project profitability. The operation of high-grading involves the removal of ore from a mine in order of descending margins of profit; since the profit margin is influenced by many factors other than grade (accessibility, mining and metallurgical recovery, mining cost, etc.), the term "high-profiting" would probably be more appropriate than "high-grading".

E.G. Thomas (1976) used a hypothetical orebody, whose distribution of values is shown in Fig. 2.8, to illustrate the benefits of high-grading. His approach also involves the initial establishment of an average mining grade (equal to or less than the average grade of the ore above cutoff, depending upon economic policy and requirements). During the first half of the mine life the ore is mined at above the average grade, and during the second half at below the average grade, grade decrease being in progressive relatively small steps from start to finish.

Assuming the mining constraints given in Fig. 2.9, the cash flows for an operation mining at the average grade (see Fig. 2.10) and for an operation in which high-grading is done (see Fig. 2.11) were calculated, and their economics compared using the discounted cash flow rate of return (DCFROR), as shown in Fig. 2.12.

| | Details of reserves in hypothetical orebody | | |
Reserve category	Tonnage	Grade range % Cu	Average grade % Cu
1	10 000 000	.0.70—0.75	0.725
2	10 000 000	0.65—0.70	0.675
3	10 000 000	0.60—0.65	0.625
4	10 000 000	0.55—0.60	0.575
5	10 000 000	0.50—0.55	0.525
6	10 000 000	0.45—0.50	0.475
7	10 000 000	0.40—0.45	0.425
8	10 000 000	0.35—0.40	0.375
9	10 000 000	0.30—0.35	0.325
10	10 000 000	0.25—0.30	0.275
11	10 000 000	0.20—0.25	0.225
12	10 000 000	0.15—0.20	0.175
13	10 000 000	0.10—0.15	0.125
14	10 000 000	0.05—0.10	0.075

FIG.2.8 (Thomas,1976)

Mining programme and related factors

1. Annual production   10 000 000 tonnes
2. Mining cost $1 per tonne of ore
3. Milling/smelting cost $2 per tonne of ore
4. Mine/mill capital cost $100 000 000
5. Copper sale price $1000 per tonne
6. Metallurgical recovery   100 per cent

FIG.2.9   (Thomas,1976)

FIG.2.10 (Thomas,1976)

Results of economic analysis based upon mining at average grade

Year	Capital charge $\times 10^6$	Operating charge $\times 10^6$	Operating returns $\times 10^6$	Overall cash inflow $\times 10^6$	Present value of overall cash inflow at 10% $\times 10^6$	at 20% $\times 10^6$
0	100	0	0	−100	−100.0000	−100.0000
1	0	30	50	20	18.1818	16.6666
2	0	30	50	20	16.5288	13.8888
3	0	30	50	20	15.0262	11.5740
4	0	30	50	20	13.6602	9.6450
5	0	30	50	20	12.4184	8.0374
6	0	30	50	20	11.2894	6.6978
7	0	30	50	20	10.2630	5.5816
8	0	30	50	20	9.3300	4.6512
9	0	30	50	20	8.4818	3.8760
10	0	30	50	20	7.7108	3.2300
					+22.8904	−16.1516

FIG.2.11 (Thomas,1976)

Results of economic analysis based upon high-grading

Year	Capital charge $\times 10^6$	Operating charge $\times 10^6$	Reserve category mined Table 1	Average mining grade % Cu	Operating returns $\times 10^6$	Overall cash inflow $\times 10^6$
0	100	0	−	−	0	−100.0
1	0	30	1	0.725	72.5	42.5
2	0	30	2	0.675	67.5	37.5
3	0	30	3	0.625	62.5	32.5
4	0	30	4	0.575	57.5	27.5
5	0	30	5	0.525	52.5	22.5
6	0	30	6	0.475	47.5	17.5
7	0	30	7	0.425	42.5	12.5
8	0	30	8	0.375	37.5	7.5
9	0	30	9	0.325	32.5	2.5
10	0	30	10	0.275	27.5	−2.5

FIG.2.12 (Thomas,1976)

Summary and comparison of results by average-grading and high-grading

	Average-grading	High-grading
Discounted cash flow return	15.1%	25.8%
Life-of-mine	10 years	10 years
Average ore grade	0.50% Cu	0.50% Cu
Total copper mined	500000 tonnes	500000 tonnes

In a "normal" situation (as above), high-grading increases the profitability, and it can be shown that its higher profitability allows (in answer to a government-enforced "socialistic approach") for a decrease in cutoff grades, resulting in increases in metal recovery and in life of the mine.

Nevertheless, as G.P. Walduck (1976) points out in his comment on Thomas' paper, the criterion of high-grading may not produce optimum benefits when cyclical price variations (similar to those in Fig. 2.7) have negative effects on the cash flow situation. Ideally, the mine should "high-profit" during the early years, and "high-grade" during the high-price cycles; being the plant capacity constant, the only way to increase the output of metal during these periods is by high-grading, assuming that the costs remain constant and high-grading does not involve increased costs that would cancel out the increase in revenues.

Furthermore, high-grading could lead to negative cash flows during the last years of the operation, as shown in Fig. 2.11, thus reducing the ore recovery and the life of the mine. (The Wealth Growth Rate, WGR, could be used here as well, to compare the profitability of these alternatives of shortened life with the profitability of mining at an average grade).

High-grading can also affect factors other than the mining costs, such as metallurgical recovery, ore availability, resource availability, plant bottlenecks, stockpiling, etc. For copper deposits it is not uncommon to have higher grades closer to the surface (due to secondary enrichment), and open-pit mines (and very often underground operations as well) will frequently be forced to follow a high-grading approach. However, in underground mines the question is seldom "now or later": more often it is "now or never".

In summary, a mining schedule that considers high-grading normally results in optimum benefits, but it must be treated carefully and adapted to the local conditions.

3. UNDERGROUND AND OPEN PIT MINING

The decision whether to mine an orebody using open pit mining or underground mining methods is taken based on a trial-and-error process, in which the profitabilities of different alternative layouts are compared. The problem is discussed in the section on optimal depth of an open pit (section 4.5), in which the Net Present Value is the economic criterion used to evaluate the different alternatives.

Since economics is the main criterion applied to choosing a certain mining method, the costs of open pit and underground mining are compared in this section, and B. Hedberg's (1981) diagnosis on the origins of the differences is summarized; valuable criteria for planning underground operations can be derived from Hedberg's paper.

In the last three decades the scale of the operations in open pit mining has increased considerably, largely because mining equipment with very high capacities became available. The development of underground equipment has not been so rapid because of the restrictions imposed by the reduced size of the openings, the size of the orebodies, shafts, inclines, drift systems, mining layouts, etc.

Because of the limited space in underground openings, when comparing operations with the same capacity, at least three times as many mobile machines have to be used underground as in open pit mining. Many small machines mean that the number of personnel in underground mining has to be much higher than in an open pit operation; on top of this, the costs for underground manhours are higher, and the time used for transportation to and from the work places reduces the manhours used for effective production.

The comparison of cost items, see Fig. 3.1, made for two large scale operations (one underground, the other open pit) under the same general conditions, shows that the cost of labour is almost five times as high underground as in an open pit, and the costs of spare parts, maintenance, fuel, etc. are about twice as expensive; this reflects the increased number of small mobile equipment in underground operations.
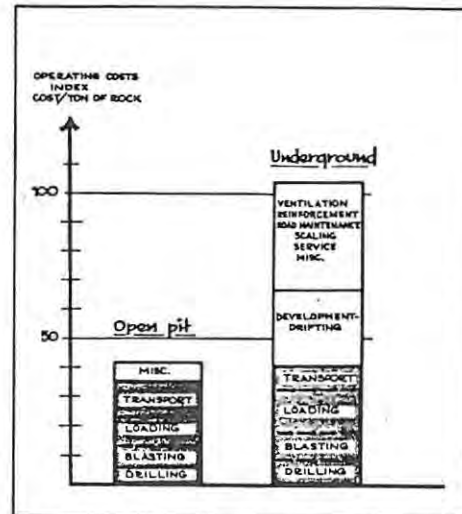
FIG.3.1 (Hedberg,1981)

FIG.3.2 (Hedberg,1981)

The comparison of unit operation costs, see Fig. 3.2, shows that there is no major difference in the costs of transportation, loading, blasting and drilling between underground and open pit mining. Furthermore, the cost of efficient underground horizontal transport on rail followed by vertical shaft hoisting (even from a considerable depth) is cheaper than the truck haulage of some kilometres in an open pit. The major difference is the amount of development works needed underground.
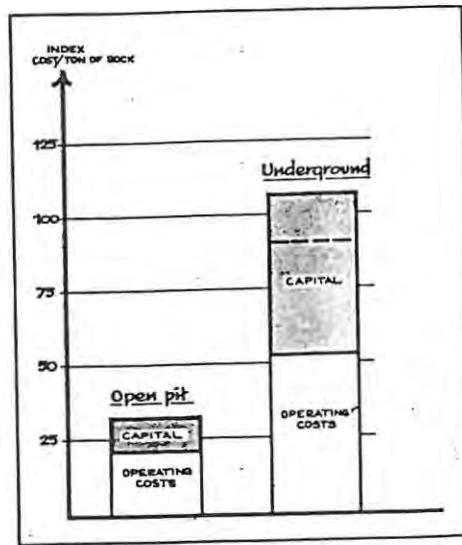
Overall
open pit and
underground
production costs
compared.

FIG.3.4
(Hedberg,
1981)

Comparison
of open pit and
underground
capital costs for
two different sized
operations.
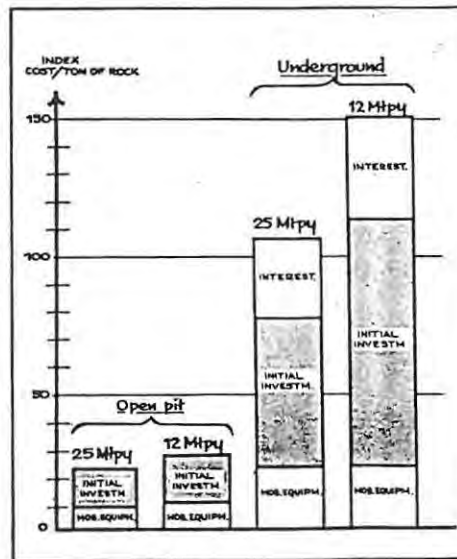
FIG.3.3 (Hedberg,1981)

Due to the number of mobile equipment units that are necessary underground, the investment costs per ton of capacity are twice as high (see Fig. 3.3). The investments in fixed production utilities are about three times higher underground, and since this investment is made much earlier than in open pit operations the interest costs are also higher (see Fig. 3.3).

Therefore, in order to optimize an underground design, thus reducing its production costs and making it competitive with open pit mining, the following factors should be considered:

(i)   increase in the knowledge about rock strength and stresses, and development of cheaper and efficient support methods (e.g. bolts and gunniting), to enlarge the size of the openings and reduce their number (because high capacity machines can be accommodated);

(ii)   development and use of high capacity machines;

(iii)   more automation of the operations;

(iv)   more effective transportation systems for personnel;

(v)   reduce the development work, applying layouts in which more ore is reached from each underground drift (Hedberg suggests the use of methods involving large diameter holes and large blasts, and requiring minimum development work); and

(vi)   reduce the overall investment, using continuous mining methods at ever increasing depth and avoiding new main levels.

4. OPEN PIT MINING

The history of surface mining is essentially that of mining coal, copper and iron ores, and the nonmetallic minerals/materials - clays, gypsum, phosphate rock, sand, gravel and stone. In this chapter the subject has been restricted to open pit mining, not considering other surface operations normally related to the mining of coal (strip mining) and placers (dredging).

The increase in labor costs and the decline in grade of the ores, resulted in a shift in emphasis from underground to surface mining during the late 1960's, but in recent years this tendency has reversed. The increase in the cost of fuel affected the costs of open pit mining to a greater extent. The saturation of the base metals market, that followed the worldwide industrial recession of the 70's and early 80's, makes more attractive the less risky small-tonnage/high-grade mining which involves a selectivity that can only be achieved by underground mining.

In recent years the trends in open pit mining have been directed towards decreasing the cost of transport. To this effect, ore-passes down to an underground primary crusher and conveyor belt transport to surface seems to be frequently used in Red China, while mobile crusher plants and conveyor belts, and trolley trucks (used even underground) are in use in other places.

In the following pages the main factors in open pit design have been approached from the point of view of an exploration geologist, who requires just a rough estimate of the mineability of a new deposit.

4.1. Manual pit design

Good descriptions on the procedures followed in manual pit design are given by Soderberg and Rausch (1968) and Koskiniemi (1977).

Compilation of basic data

The first step in the manual design of an open pit consists in the compilation of all the basic data, including the geology and the distribution of the mineralization and grades.

Geology and grades are normally plotted, interpreted and
estimated on section plans perpendicular to the direction of maximum
isotropy in the distribution of the particular feature.   These sections
are often vertical ones, but for efficient manual design of an open
pit operation, this information should be reduced to horizontal sections
corresponding to the chosen level or bench spacing.   The level or
bench spacing is a function of the angle of slope stability and the
selection of equipment (see section on blasting), which, in turn, is
related to the rate of production and the size of the operation.

Mid-bench pit rim

Mined out

——— Geological boundary	1. Shale with grit type 3
——— Pit rim	2. Shale with grit type 2
▬▬ Orebody	3. Shale with grit type 1
——— Fault	4. Chingola Dolomite
	5. Dolomitic schist
	6. Upper Banded Shale
	7. The Felspathic Quartzite

8. Banded Sandstone Lower
9. Pink Quartzite and shale Marker
10. Banded Sandstone Lower
11. Lower Banded Shale
12. Arkose
13. Granite

0 100 200 300 400 500 m

FIG.4.1
(Mason &
Lea,1979)

Portion of typical bench plan from Nchanga open-pit

In plan, most open pits must close in circular configuration,
which cannot be properly visualized nor computed from a series of
parallel vertical sections.   Vertical sections can be and are used
to approximate the fixing of pit limits (as shown further on), but
horizontal sections must be used to optimize a pit design.

An obvious exception is an open pit operation of a long orebody
outcropping along a ridge.   These pits are, in effect, long trenches
and cross sections provide the best method for designing optimum pit
limits.

The geology and distribution of ore and grades is obtained from borehole and any other source of information available. Data is interpolated, or extrapolated, onto the bench level according to the normal practice of projecting along the dip or the plunge of the structures.

In computer design of open pits the orebodies are subdivided into blocks representing a size that can be identified and mined separately, and composed of homogeneous rock and ore types. The blocks should have a depth equal to the bench height, and lateral dimensions equal to a number of times the burden and spacing of the blastholes. Block size is an important factor, and it is necessary to balance the increased detail given by a small block size and the increased efficiency of computing when handling fewer blocks of a larger size. Normally the block size is the same used by the kriging program for geostatistical estimation of the ore reserves.

The purpose of the use of blocks in computer design is to convert the basic data into a three-dimensional matrix that can be handled by computer. Normally, the use of blocks is not practical in manual design: the information of 41 bench plans at Nchanga (Zambia) has been subdivided in more than 7 000 000 blocks of 10 m x 5 m x 11 m dimensions, that can be handled only by a computer. To reduce the number of blocks for manual calculations generally involves oversimplifications.

Stripping ratio

The development of a manual pit design requires the establishment of a "break-even stripping ratio", that is applied only at the surface of the final pit, or to the last increment before reaching this final slope. This ratio is always less than the overall stripping ratio, otherwise there would be no profit to the operation, and can be determined by the formula:

$$\text{Break-even stripping ratio} = \frac{(\text{Recoverable value x ton ore}) - (\text{Production cost x ton ore})}{(\text{Stripping cost x ton waste})}$$

or including a minimum profit factor:

$$\text{BESR} = \frac{\begin{pmatrix}\text{Recoverable value} \\ \text{x ton ore}\end{pmatrix} - \left[\begin{pmatrix}\text{Production cost} \\ \text{x ton ore}\end{pmatrix} + \begin{pmatrix}\text{Minimum profit} \\ \text{x ton ore}\end{pmatrix}\right]}{\text{(Stripping cost x ton waste)}}$$

where "production cost" is the total of all costs through to the refined metal, exclusive of stripping cost.

More sophistication can be obtained in these calculations using computers, including such items as (i) production costs by levels, to reflect costs according to the length of ore and waste hauls, and (ii) plant treatment costs related to the grade of the ore because of the varying percentage recoveries.

Ratios must be developed for variations in the grade of ore and market price of the end product. The alternatives shown in Figs. 4.2 and 4.3 correspond to an example used by Soderberg and Rausch (1968) to illustrate the procedure for pit design.

CALCULATION BREAK-EVEN STRIPPING RATIO

Head (% Cu)	0.8		0.7		0.6		0.5		0.4	
Recov. Cu/ton	14.1 lb		12.2 lb		10.3 lb		8.4 lb		6.6 lb	
	Ton Ore	Lb Cu	Ton Ore	Lb Cu	Ton Ore	Lb Cu	Ton Ore	Lb Cu	Ton Ore	Lb Cu
Costs:										
[1] Mining	$0.45									
Milling Depr. } Gen'l.	1.25									
	$1.70	12.1¢	$1.70	13.9¢	$1.70	16.5¢	$1.70	20.2¢	$1.70	25.8¢
Treatment, etc.	0.85	6.0	0.76	6.2	0.65	6.3	0.55	6.5	0.44	6.7
Total cost	$2.55	18.1¢	$2.46	20.1¢	$2.35	22.8¢	$2.25	26.7¢	$2.14	32.5¢
Break-even stripping ratio:										
@ 25¢ Cu:										
Value	$3.53		$3.05		$2.58		$2.10		$1.65	
Net	0.98		0.59		0.23		[0.15]		[0.49]	
[2] Ratio	2.5:1		1.5:1		0.6:1		—		—	
@ 30¢ Cu:										
Value	$4.23		$3.66		$3.09		$2.52		$1.98	
Net	1.68		1.20		0.74		0.27		[0.16]	
[2] Ratio	4.2:1		3.0:1		1.8:1		0.7:1		—	
@ 35¢ Cu:										
Value	$4.94		$4.27		$3.61		$2.94		$2.31	
Net	2.39		1.81		1.26		0.68		0.17	
[2] Ratio	6.0:1		4.5:1		3.2:1		1.7:1		0.4:1	

[1] Excluding stripping cost
[2] At stripping cost of 40¢ per ton waste
( ) Indicates negative figure

(Soderberg & Rausch, 1968)

FIG. 4.2
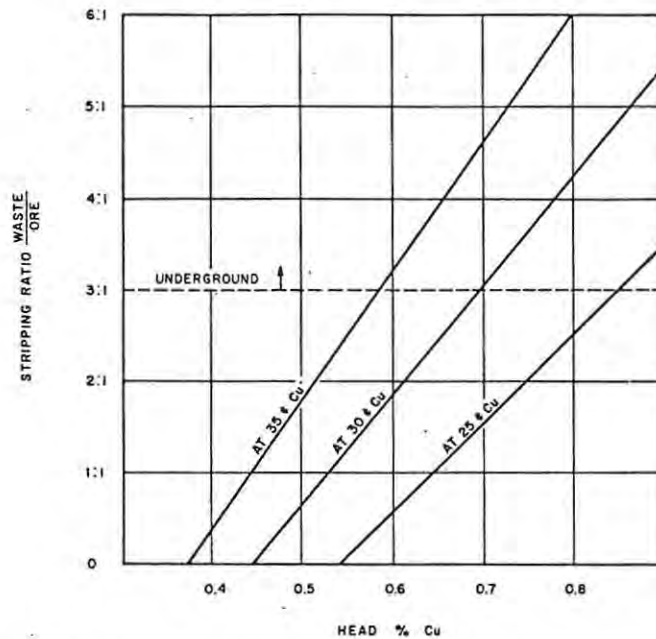
FIG. 4.3 Break even stripping ratio related to grade of ore and market price. (Soderberg & Rausch, 1968)

Care should be taken in the use of kinds of costs. It is obvious that all the direct costs should be used. Depreciation cost should also be included, although it may be assumed that this has been fully recovered by the time the final pit limits are reached. Capital costs (preproduction and financing costs) should not be included, because such costs are recovered before the end of the life of the mine.

Ultimate pit slope

After estimating the break-even stripping ratios, the final pit slope must be determined. To minimize the overall stripping ratio, the slope should be as steep as possible and still remain stable. In other sections of this chapter the geological factors affecting the slope stability are discussed in detail, same as the interrelationships between the ultimate pit slope and the bench slope, the bench height, and the berm width.

The pit should be designed to a conservative final-slope angle, and this final slope fixed at the break-even stripping ratio for the last increment.   An interesting fact is derived from the application of the break-even stripping ratio to the degree of slope: the surface intercept of the pit limit is often the same for varying degrees of slope, as it is shown in Fig. 4.4.
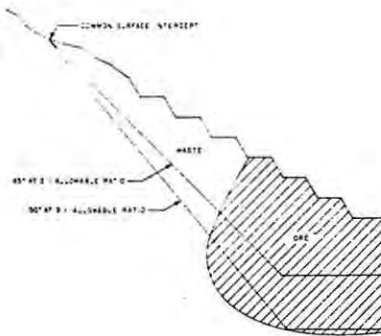
FIG.4.4 Common surface intercept of two different pit slopes at the same allowable stripping ratio.
(Soderberg & Rausch,1968)

Grade cut-off

The cut-off grades have been discussed in some detail in a previous chapter.   Its determination involves more than the simple calculation of a break-even grade, or "pay limit", as shown in Fig. 4.5.   In this example a 0,437% Cu cut-off cannot carry the cost of any stripping. The "cut-off grade" for the purposes of this method of pit design does not include the costs of stripping in its calculation, since the stripping ratio is going to be used as a variable.

Head	0.50% Cu		0.40% Cu	
Recoverable copper per ton	8.4 lb		6.6 lb	
Costs	Per ton Ore	Per lb Cu	Per ton Ore	Per lb Cu
Mining	$0.45		$0.45	
Milling, general expense & depreciation	1.25		1.25	
	$1.70	20.2¢	$1.70	15.8¢
Treatment, selling & delivery	0.55	6.5	0.44	6.7
Total production cost excluding stripping	$2.25	26.7¢	$2.14	32.5¢
Value @ 30¢ market price for copper	2.52	30.0	1.98	30.0
Net value per ton	$0.27	3.3¢	($0.16)	(2.5¢)
Indicated grade cutoff (by interpolation)		0.437% Cu		

(Soderberg & Rausch,1968)   FIG.4.5

Pit design

After establishing the limiting stripping ratios, the ultimate degree of pit slopes, and the grade cut-off, the project is ready for the pit design. The pit should be designed on a conservative basis, since market prices and costs change with time and are important factors in fixing the final pit limits. The pit can always be expanded should future prices and costs justify the change.

The design is developed from vertical sections on which the surface topography and the ore limits, out to the adopted "grade cut-off", are shown. To begin with, the sections should be spaced at regular intervals; they should be parallel to each other and normal to the long axis of the orebody (see Figs. 4.6 and 4.7).
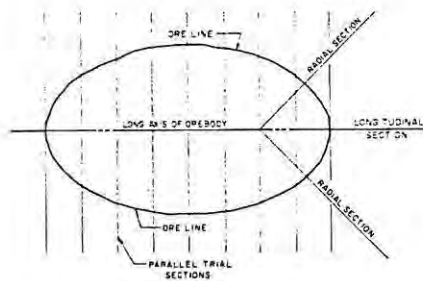
FIG.4.6   Plan of orebody.   FIG.4.7   Typical parallel trial section.
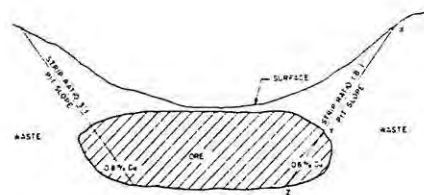
(Soderberg & Rausch,1968)

The process of finding the pit limit is one of trial-and-error, moving the pit slope (at the adopted angle) from the center towards the periphery of the orebody in successive increments, and calculating the ore grade and the stripping ratio for each increment. The final pit slope is reached when the stripping ratio for the last increment corresponds to the break-even ratio for the grade of the ore in that increment, as derived from graphs similar to the one shown in Fig. 4.3.

In Fig. 4.7 the pit slope line XYZ is fixed at the point that results in a stripping ratio of 1,8 waste : 1 ore, according to the proportional lengths of intercepts:

$$\frac{\text{Length of XY waste}}{\text{Length of YZ ore}}$$

(in more complex geological situations, it is better to work with
increments and obtain the stripping ratio as the ratio between the
areas of waste and ore measured with a planimeter). This is the
economic ratio for the indicated ore grade of 0,6% Cu in this area,
as taken from the graph in Fig. 4.3 at a 30 ¢ market price for the
pound of copper.

For the other side of the section the indicated grade of the
orebody is 0,8% Cu, and the same graph in Fig. 4.3 gives an allowable
stripping ratio of 4,2:1 at the 30 ¢ copper price. However, a
comparison of open pit and underground mining costs has given a break-
even at, say, a stripping ratio of 3:1. Therefore, the pit slope
limit is fixed at the 3:1 stripping ratio; beyond that point, under-
ground mining is relatively more profitable in spite of open pit
mining being still economic.

In the last situation, when comparing underground and open pit
mining costs, the costs of underground mining are going to be burdened
by capital investment in a complete new selection of equipment and in
development of a rather small tonnage.

Each trial section is similarly analyzed, and the surface intercepts
of the pit limit, together with the ore intercepts at the pit limit and
the bottom toe of the pit limit, are all plotted on a plan map.

Radial sections, showing more accurately the actual stripping
ratios, must be used to fix the limits at the pit ends (see Fig. 4.6),
or in any other place where the shape of the orebodies forces the pit
design to curve. In these places, the pit slope limiting line must
be fixed at stripping ratios considerably under the indicated allowable
ratio to give effect to the curved-end geometry of the pit.

The intersections of the pit slope limit lines with the horizontal bench levels are plotted onto the composite horizontal plan, usually resulting in an uneven geometric pattern. This must be evened out, which means that the indicated ratios on the trial sections will be adjusted to plus and minus values, but averaged to its economic limit.

The resulting pit design is then checked in plan as illustrated in Fig. 4.8. The true stripping ratios by sectors are indicated by relative areas of waste and ore, which are readily determined by planimetering; it must be kept in mind that these stripping ratios are at the final pit surface and do not reflect the overall stripping ratio, which for the previous example would be about 1.5 (or 2:1). The grade of ore at the pit limit shown in the plan, is obtained from the vertical sections or from horizontal sections corresponding to each bench.

The allowable limiting stripping ratios related to the sector ore grades, are determined applying the break-even stripping formula, at the 30 ¢ market price for copper (for this example), to the new set of grades of ore at the pit limit. The results are shown in Fig. 4.9. Sectors 3 and 4 permit break-even stripping ratios of 4.2 and 3.6 respectively. Under the hypothetical assumption that the break-even cost with underground mining is at a stripping ratio of 3:1 (representing a total open pit mining cost of $0.45 + (3 x $0.40) = $1.65 per ton of ore, the maximum ratio is fixed at 3:1 for sectors 3 and 4, as shown in Fig. 4.8.

Averaging of ore grades and stripping ratios is not a sound technique for pit design. Each sector must carry its economical ratio.

It is apparent that this method involves an iterative "trial-and-error" process to finalize pit designs, and computer programs have been developed to design pits producing print-outs of pit limits for each bench. Multiple pit designs can be determined by considering changes in design criteria or changes in specifications such as slope angle,

PLAN OF ULTIMATE PIT   SECTION A—A

(Soderberg & Rausch,1968)   FIG.4.8

Sector	Head % Cu	Recov. Cu/ton	Value @ 30¢ Cu	Production Cost/ton	Available for Stripping	Break-even Stripping Ratio*
1	0.62	10.7 lb	$3.21	$2.37	$0.84	2.1:1
2	0.60	10.3 lb	3.09	2.35	0.74	1.8:1
3	0.80	14.1 lb	4.23	2.55	1.68	4.2:1
4	0.75	13.1 lb	3.93	2.51	1.42	3.6:1
5	0.70	12.2 lb	3.66	2.46	1.20	3.0:1
6	0.57	9.7 lb	2.91	2.30	0.61	1.5:1
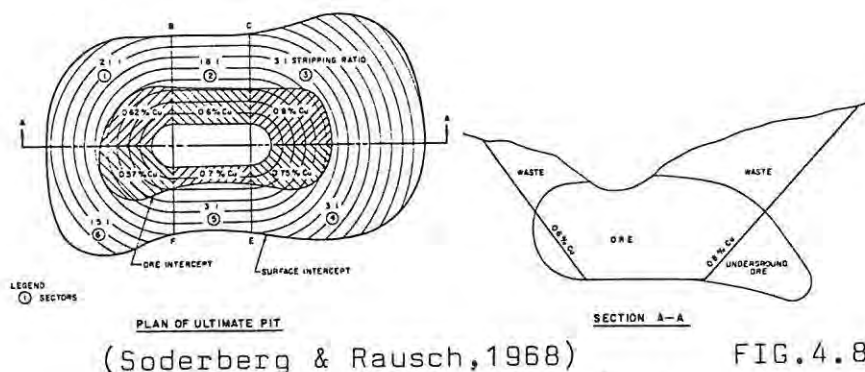
Allowable Break-even Stripping Ratios

* @ 40¢ per ton stripping.

(Soderberg & Rausch,1968)   FIG.4.9

limiting stripping ratios under various market values, and grade
cut-offs. Different slope angles can be used in the various sections
according to the properties of the local rocks. Minimum profit
requirements are also readily handled through the break-even stripping
ratio.

The use of a break-even stripping ratio as a criterion to
determine the ultimate pit slope works very well for a simple configura-
tion like the one used to illustrate the technique. However, more
complex geological situations, such as the Nchanga copper mine in
Zambia (see Fig. 4.10), are often found and the method must be adapted

accordingly.   In the slightly more complex situation shown in
Fig. 4.11, the increment I_1 could have reached the break-even
stripping ratio, but if we continue mining to include the second
orebody, or alternatively the second plus the first, the increments
I_2 and I_3 could be well under the break-even stripping ratio.

In this case the criterion should be used to delineate some
alternatives, say the pits 1, 2 and 3 (Fig. 4.11) in which, for
example, the increments I_4 and I_5 would have stripping ratios equal
to the corresponding break-even ones.   To choose among them, a
mining sequence should be derived from each alternative and economic
criteria, such as the rate of return and the net present value, used.
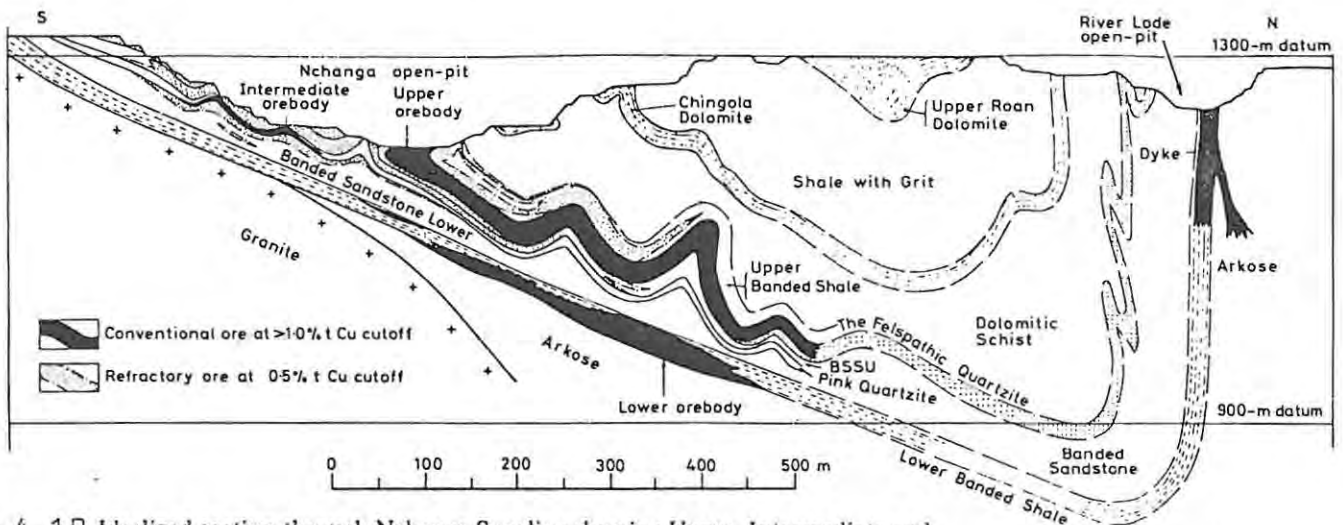
FIG.4.10 Idealized section through Nchanga Syncline showing Upper, Intermediate and
Lower orebodies
(Mason & Lea,1979)
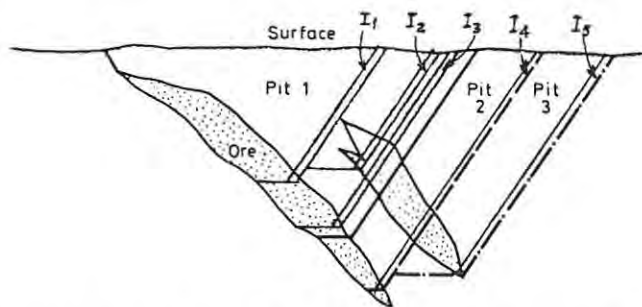
FIG.4.11 Section through hypothetical orebody showing
first pit mined and subsequent increments
(Mason & Lea,1979)

The Nchanga pit was designed (Mason and Lea, 1979) using a simple computer program which starts with the input of a manually submitted pit-bottom perimeter. Several alternative pits were designed from a series of pit-bottoms, each 40 m wide (the minimum efficient operating width of the available equipment) and 200 m long. These alternative pits are shown in Fig. 4.12, and they could perfectly represent increments in a manual trial-and-error process of determining the final pit design, according to the technique previously mentioned.
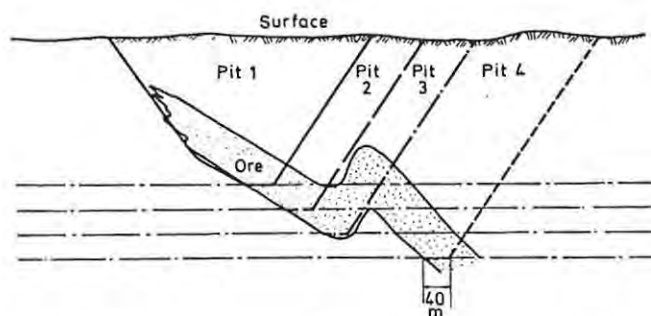
FIG. 4.12   Simplified section through Nchanga Upper ore-body showing layout and inter-relationship of standardized pit bottoms   (Mason & Lea, 1979)

Incorporation of access roads

Taylor (1971) developed a computer program to incorporate the access roads into computer-generated open pits. The principles involved can be used in a manual design of access roads in a manually obtained open pit configuration. The optimum road design is important since the haulage costs often constitute a very high proportion (30 to 50%) of the total mining costs. The development of large-tonnage, low-grade open pits makes the economics involved in pit design more critical, and the overall effect of the incorporation of access roads is often important even in the early stages of feasibility studies.

Important factors in the design of access roads are:

(i)   The minimization of the transport costs of ore and waste from the open pit throughout the life of the

mine, which for a final pit made up of a series of blocks or unit volumes Vi can be stated as:

$$\min_{i=1,\,N} (M_oC_od_i + M_wC_wS_i)$$

FIG.4.13
(Taylor,1971)

where

C_o = cost of ore haulage per unit distance in £/ton
C_w = cost of waste haulage per unit distance in £/ton
M_o = tonnage of ore in unit volume V_i
M_w = tonnage of waste in unit volume V_i
d_i = haul distance of ore from block V_i to concentrator
S_i = haul distance of waste from block V_i to waste dump
N = number of blocks within the pit.

(ii)   The desire to mantain permanent access roads rather than temporary ones. This allows electrification for trolley-trucking, asphalt or crude oil surfacing reducing tire wear, and avoids having to allocate equipment to developing new roads.

(iii) The avoidance of areas of inherent slope instability within the pit.

(iv)   The avoidance or minimization of traffic congestion within the pit.

The location of the concentrator and waste dump is often fixed by topographic considerations or, in an optimum situation, have been fixed after reference to a preliminary pit design. The initial alignment of the road is determined by the position of the concentrator and the waste dump, and the distribution of ore and waste in early stages of production, so as to satisfy the condition stated in Fig. 4.13.

Most of the development work to date has assumed that the haul roads spiral around the pit until the lowest bench is reached, but the spiral layout cannot be used in some sections and an alternative zig-zag layout is preferred. In very large pits the haul distance may be too long using the spiral layout, whereas the zig-zag can keep the roads closer to the production-gravity center of the pit. Sections of the pit slope may be relatively unstable and the form of haul road must avoid these areas. In stratiform orebodies, permanent haul roads generally zig-zag down the footwall side of the pit.

In contrast, spiral is preferred over zig-zag when the pit walls are too steep to allow the minimum radius on the bends without removal of large quantities of waste from the pit slopes.

There are two possible layouts for the access roads, which are shown in Fig. 4.14 :   increasing the stripping ratio and keeping the ore recovery constant (Figs. a), or moderately increasing the stripping ratio by reducing the ore recovery (Figs. b).
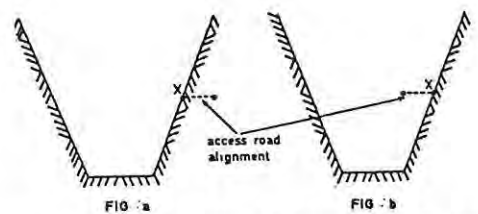
Cross sections of an open pit showing two possible modes of incorporating a haulage road.
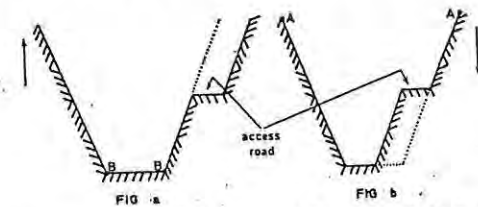
Effect on over-all pit design of the two haul-road layouts shown in Figures 4a and 4b.

FIG.4.14   (Taylor,1971)

When mining vertical orebodies, the roads can be developed on any side of the pit with similar effects on the stripping ratio and ore recovery.   On higher levels the roads are cut in the way shown in Fig. 4.14 a, maintaining the same ore recovery at a low increase in the stripping ratio.   The roads are cut in this way until the incremented stripping ratio gets close to the break-even point. From there on (i.e. in the lower levels) the roads are cut in the way shown in Fig. 4.14 b, resulting in a lower rate of increase in the stripping ratio, this time produced by moderate losses in ore. At the bottom of the pit the modified overall stripping ratio (at the ultimate pit slope) should be the break-even one or less.

The place to change from one system of cutting the road to the other is obtained by a trial-and-error process.

In non-vertical orebodies the method of cutting a road shown in Fig. 4.14 b produces   less ore losses (and consequent increases in the stripping ratio)   when used on the hangingwall side of the body.

The procedure to incorporate the access road is as follows:

(i)   Locate the points of intersection of the initial
       access road alignment with the uppermost perimeter
       of the pit : the point B in Fig. 4.16.

(ii)   Determine the slope length, l, of an access road
       inclined at a maximum permissible grade, g, between
       levels i and i+1 (Fig. 4.15).

$$l = \frac{m}{\sin \theta}$$ where m = specified bench height
                                       θ = arc tan $\frac{g}{100}$

                                where g = access road grade (%)

(Taylor,1971)   FIG.4.15

FIG.4.16 (Taylor,1971)
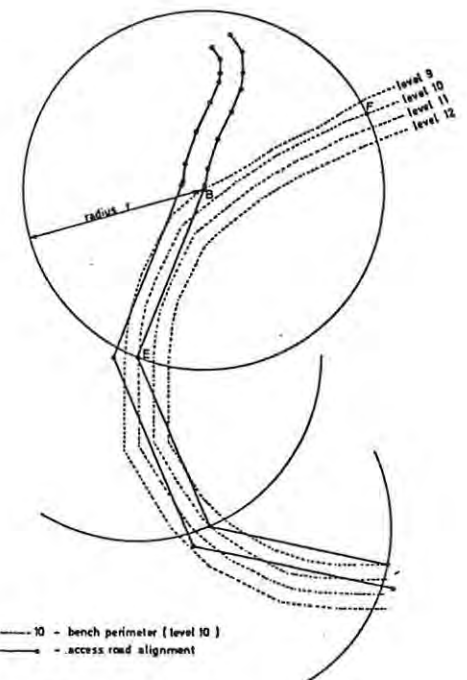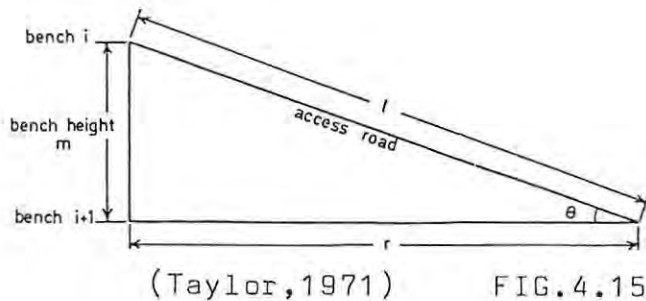
(iii)   Having the point B as the center determine the intersection
of a circle of radius   r = l cos θ with the perimeter
of bench   i + 1 (see Fig. 4.16).

(iv)   Using the appropriate point on the perimeter of bench
i + 1, repeat the procedure outlined in steps (ii) and
(iii).   This is repeated until the perimeter of the
lowest bench is reached, thus generating a series of
points that joined by straight lines (see Fig. 4.18)
produce a rough shape of access road.

(v)   Fig. 4.19 shows how the neighbouring perimeter contours
of the different benches can be used to smooth the
alignment of the road.

(vi)   Having decided which of the two possible layouts
(Fig. 4.14 a or b) is required, the second margin of
the road is generated parallel to and to one of the
sides (according to the choice of layout) of the road
alignment curve produced above.   This second curve is
separated from the first by a distance equal to the
access road width.

(vii)   The original pit is redesigned, now including the
road.
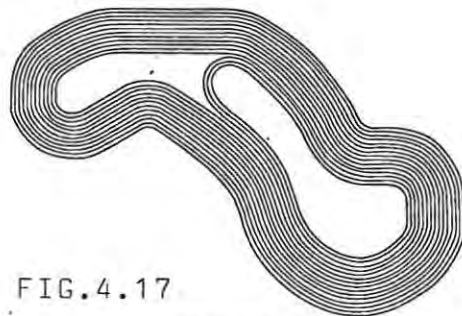
FIG.4.17

— Preliminary open-pit configuration.
(Input data for access road design program)
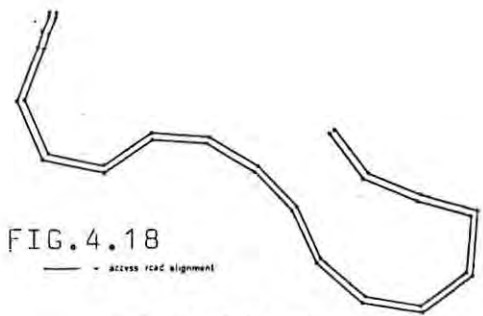
FIG.4.18

——— access road alignment

— Access road alignment.
(Prior to smoothing procedure)

FIG.4.19

— bench perimeter (level 10)
——— access road alignment
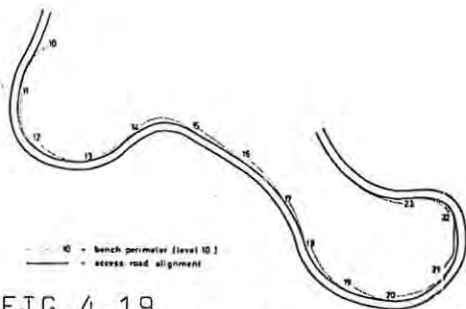
— Smoothed access road alignment.
(Showing sections of adjacent bench perimeter contours
used in the smoothing procedure)
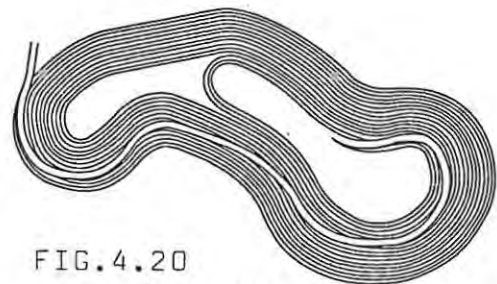
FIG.4.20

— Final open-pit configuration.
(Showing incorporated access road)

(Taylor,1971)

4.2. Computer generated open pits

Several companies are developing interactive computer graphic capabilities, which will allow a proposed pit design to be displayed on a screen and modified at will. When such interactive systems become fully operational, they may well replace the fully automated computer methods now in use.

The fully automated pit programs work with the grade, rock-type, rock mechanics, etc., information stored in terms of blocks in a three-dimensional matrix.

Of these fully automated methods, the so-called "moving cone" method has found the widest acceptance. In this technique, an inverted conical frustum having a side slope equal to the projected side slopes of the ultimate pit is "superimposed" on a three-dimensional block model of the deposit. The cone "moves" through the model, examining all combinations of blocks meeting specified criteria to determine the one configuration that provides the optimum economic solution. Such solutions tend to be very costly and generate mixed results. If the problems are sufficiently constrained to obtain a truly optimal solution, computer costs are likely to become exorbitant; if unconstrained (and therefore less expensive) executions are done, the real optimum pit limits will probably not be determined.

In the future, an unconstrained system will probably provide a preliminar pit configuration that will be subsequently optimized making use of one of the above mentioned interactive systems.

4.3. Mining sequence

After designing a possible open pit, it is necessary to establish a rough sequence of mining in order to assess the economic feasibility of the design.

Some general factors will affect this sequence. Mining proceeds from top to bottom benches maintaining a stable working slope. In each

bench, mining starts next to the access road retreating in a direction dictated by the grade-control practice adopted (discussed in the section on grade control in this same chapter) and in particular by the decision of high-grading or not.

The decision to high-grade during the early life of the mine, or to mine close to the average grade of the deposit throughout the mine life, must be made before the actual planning is started. High-grading is very often constrained by mining considerations, and the most simple way of generating positive cash flow during the early life of the mine, and keeping a plant head-grade relatively constant, is by minimizing the high stripping ratios of those early years. The slopes should be as steep as possible and, at the same time, provide bench room for optimum operating efficiency.

Fig. 4.21 shows the relationship of equipment size, bench height and operating room when a 15- cu.yd. shovel is loading into 85-ton trucks. The ample working room required by the equipment for operating efficiency means flat working slopes in contrast with the steepened final slope (see Fig. 4.22), and stripping ratios in early mining stages are much higher than the overall ratio.
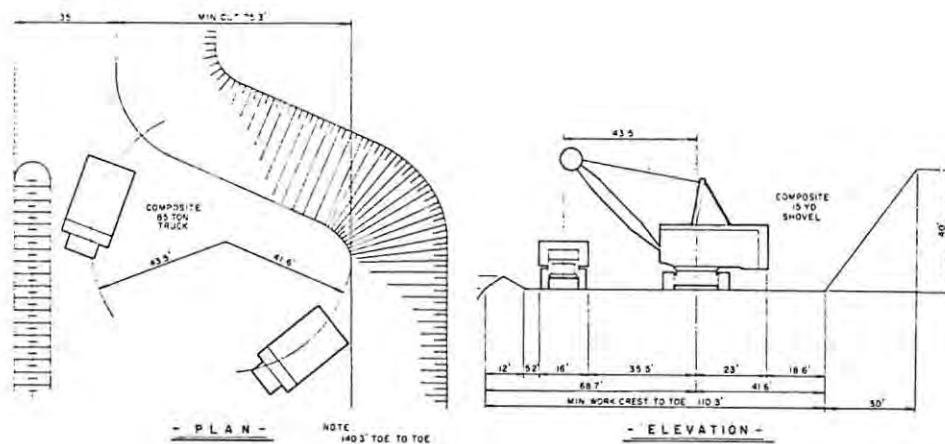
FIG.4.21   Relationship of equipment size, bench spacing and operating room.

(Soderberg & Rausch,1968)
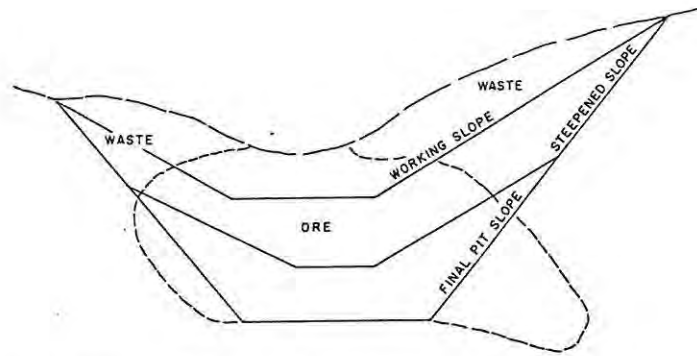
FIG.4.22 . Relation of working slope to final pit slope.
(Soderberg & Rausch,1968)

To minimize the high stripping ratios during the early years,
mining should start close to the footwall side of dipping bodies
(see Fig. 4.24), or close to the center of vertical bodies.
The working slopes can be made steeper by mining several benches as
a group, as it is shown in Fig. 4.23 where the grouping of five
40' benches into one shovel operating unit — in which the shovel
moves from bench to bench — gives a working slope steeper than
24° 30'; working each bench as an independent shovel operating
unit, each requiring enough operating room, the working slope would
have been about 15° for the same example.
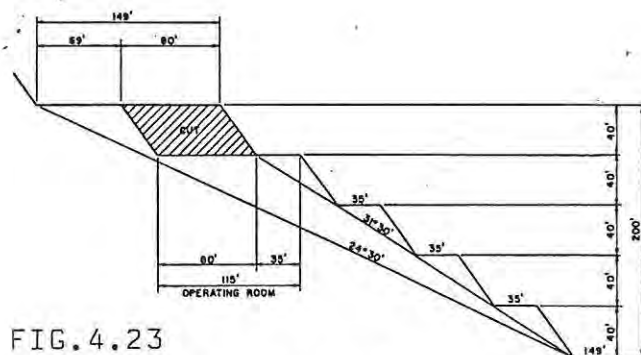
FIG.4.23

. Method of increasing the angle of the working slope by mining
several levels as a group. (Soderberg & Rausch,1968)

If flat working slopes result in high stripping ratios in early
years of operation, once the working slopes reach the final pit
surface limits the stripping ratio decreases considerably (see Figs. 4.22

and 4.25). The planning of stripping sequences must avoid peak
ratios for short time periods, that means high equipment investment
for the peak period followed by reduced requirement and resulting in
poor utilization.

In a manual procedure of planning the mining sequence, the bench
plans generated during the pit design stage are divided into blocks.
The blocks can be larger in size than those used for computer pit
design, since we want the planning of annual productions and, on the
other hand, only a rough plan of the mining sequence is needed to
obtain the preliminary estimate of the economic feasibility of the
project we are interested in.

Each block is classified as ore or waste, and is given a tonnage
and grade of ore (recoverable using good grade control practice) and
a tonnage of waste. The block is then given a mining priority as
shown by the consecutive block numbers in the section in Fig. 4.24.
The priorities are given in accordance with the criteria previously
mentioned; for example, in this dipping tabular orebody (Fig. 4.24)
mining begins close to the footwall side, where the stripping ratio is
lower and a permanent access road can be cut at a much lower stripping
increase, and tries to reach the final pit wall on the footwall side
as fast as possible. It is obvious that when planning the mining
sequence vertical cross-sections can give a good idea of the stripping
sequence, which must be optimized in the way shown in Fig. 4.25 (low
rate of stripping in early years and trying to avoid sudden peak ratios).

FIG.4.24   (Nilsson & Burgher,1982)
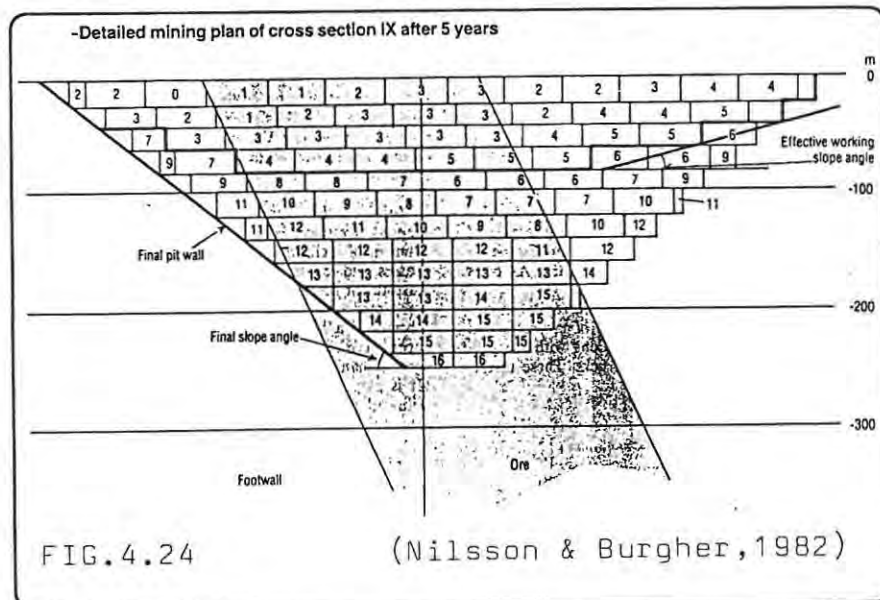
Once a mining rate is fixed a detailed mining plan, such as the
one in Fig. 4.25, can be obtained.   The mining plan shown in Fig. 4.25
corresponds to the pit alternative of Fig. 4.24, for a mining rate of
5 mill. tons of ore per year (Nilsson and Burgher, 1982).   Note that the
mining plan not only shows the tonnage of ore and overburden-waste
mined annually, but also the amount of each of those materials
coming from different benches.   This last information is going to be
important in the economical appraisal of the project because the
transport, a very important cost item, is related to the depth or
distance of transport.   As well as the tonnage of waste and ore,
the average grade of the ore mined every year can be estimated
from the mining sequence and from the grade of the blocks mined.
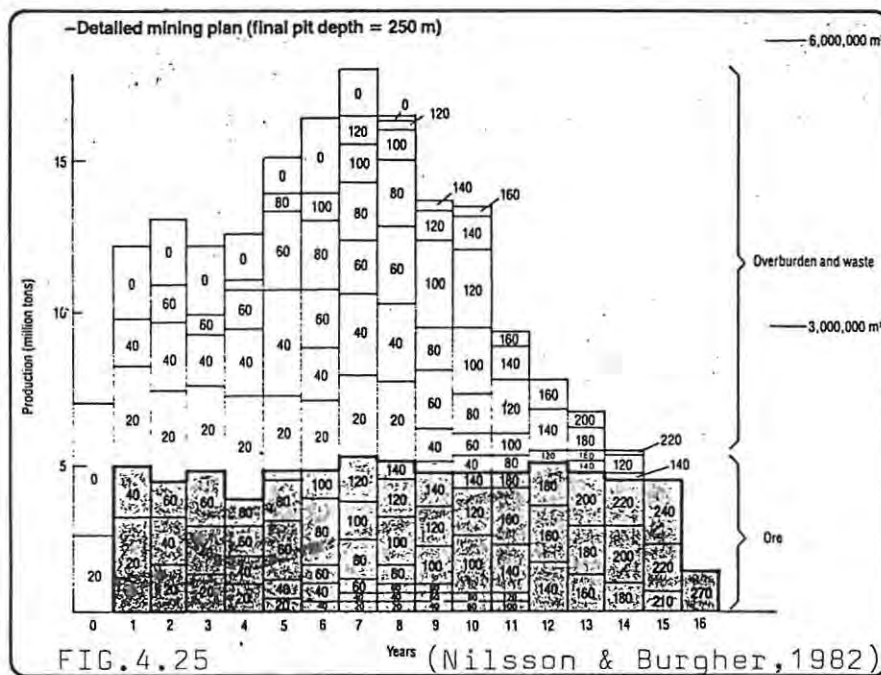
FIG.4.25   Years (Nilsson & Burgher,1982)

4.4. Economics

When the detailed mining plan has been established, the annual
cash flows can be estimated and Discounted Cash Flow techniques used
to evaluate the profitability of the project.

Fixed operating costs can be given to each ton of broken ore/waste or to each block used in the planning of the mining sequence (see Fig. 4.24). The two main variable operating costs are the depth-related (transport) and the rock-type related (drilling and blasting, and haulage-plant treatment- etc., if we consider waste and ore as being different "rock types"). Variable operating costs can be given to each block according to its "rock type" and to the depth of the block, or distance of transport of its material to the plant or to the waste dump.

Revenues can be estimated block by block as well, and they result from the grade, tonnage, mineralogy, and distribution of the ore in each block. The mineralogy affects plant recoveries, grade of the concentrates and, consequently, the cost of transport of those concentrates and the smelting charges. The distribution of the ore within the block affects the mine recovery and the dilution (some discussion can be found in the section on grade control).

The capital costs have a fixed component (initial investment in equipment, infrastructure, etc.) and a time-variable one : stripping. The variation of the second one can be deduced from the mining plans.

When the costs and revenues associated to every block are known, it is easy to calculate the annual cash flows for a certain mining plan, or even to combine those values in different ways in order to evaluate alternative pit configurations. At Nchanga, a computer program estimated the net profit for every block in which the mine was divided, including the cost of stripping for that particular block; tables similar to the one shown in Fig. 4.26 were drawn and the economic pit limits derived by summing the values (net profits) from the top of each column downwards until a maximum was reached (Mason and Lea, 1978).

A procedure similar to the one used at Nchanga could, perhaps, be used manually. The problem is how to calculate the stripping ratio associated to each block, and how to include the stripping cost

SECTION	101	103	105	107	109	111	113	115	117	119	121	123	125	127	129	131	133	135	137	139	141	143	SECTION
BENCH																							BENCH
85																		+3 03	+0 7x	+0 x3	-0 5x		85
107																		+x 77	+5 33	+x 57	+0 99		107
129								AREA ALREADY EXHAUSTED										+2 x8	+5 7x	+x 0x	+1 57		129
151																	+0 1x	+2 1x	+x x5	+x 37	0 xx		151
173																	+1 x2	+1 x5	+2 x2	+0 77			173
195																+1 35	-2 xx	-0 0x	+1 53	-0 1x			195
217															+0 75	+x xx	-2 x5	-0 xx	-0 12	-2 0x			217
239			+2 90	-0 79							+x 79	+2 33	+1 x5	+3 07	+x x2	+9 70	-0 x3	-0 xx	+0 12	+0 0x			239
261			-1 xx	+x 7x						+7 x9	+x x0	+2 27	+x x0	+3 1x	+x x1	-5 0x	+0 15	-1 0x	+0 53				261
283			-0 79	+x 53						+x x9	+x 5x	+x 53	+x x7	+x 31	-0 0x	+1 xx	-1 3x						283
305			-2 2x	-7 xx	-x 37	+2 12	+2 xx	+x 71	+19 55	+17 5x	+5 57	+1 79	+9 0x	+2 55	-1 95								305
327			-0 0x	+2 57	+2 7x	+2 7x	+7 1x	+10 25	+10 2x	+3 x5	+1 17	-2 09	+12 09	-0 x9	+0 30								327
349			-5 0x	-0x7	+1 57	+x 5x	+5 7x	+7 07	+2 x7	-0 x0	-2 7x	+x 03	+x 93	+x x7									349
371				+x x7	+x x5	+2 1x	+1 x2	+x xx	+2 9x	+5 x7	+7 x2	+1 1x	+2 50	+x x9									371
393				-3 x2	-0 x0	+1 72	+2 2x	+7 x3	-2 7x	-x xx	-2 5x	+1 15	-1 21	+x 51									393
415				-0 xx	-1 x2	+0 9x	+0 9x	-2 05	-x 95	-5 25	-7 x3	-x 90	-3 99	-x3 11			OUTLINE OF ECONOMIC PIT LIMIT						415
437				-0 5x	-x 3x																		437
459																							459

FIG.4.26   Relative values of Upper orebody blocks at K1100/t Cu showing economic pit outline (K1100 = £780 = $U.S.1400)   (Mason & Lea,1979)

in every block.   Lines with a slope equal to the ultimate pit slope could move through the sections, from the center of the orebody to its periphery (see Fig. 4.27), at increments equal to the block widths. In that way we could obtain the amount of waste W that is necessary to be stripped out in order to mine the ore in blocks 0_1, 0_2, 0_3, etc. The stripping cost of W should be charged to the cost of mining the ore block 0_1, bringing the cost forwards in time — at an interest rate equal to the cost of capital for that company — from the year in which W was mined to the year in which 0_1 is scheduled to be mined. The decision of mining or not 0_2, and subsequently 0_3, 0_4, etc., is not affected by the cost of stripping W which was carried by 0_1 (already mined out).   In this way, net profits for every block could be tabulated and used in a way similar to the one shown in Fig. 4.26, taking into consideration that operating costs, annual capital costs and revenues are in constant money terms.   Fixed capital investment during the preproduction stage is depreciated in the early years of production.
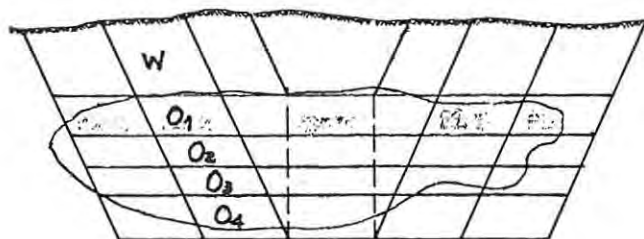
Longitudinal section through typical sequence of pits

FIG.4.27   (Mason & Lea,1979)

If we keep the rate of production fixed, the economics related to different cut-off grades can be compared and the best alternative chosen. The example shown in Fig. 4.28 was taken from Soderberg and Rausch (1968) and, in spite of its obsolete costs, it illustrates how the best cut-off grade for a fixed plant capacity of 5 mill. tons per year can be selected in a given deposit. A 0.5% Cu cut-off grade means the addition of 40 mill. tons @ 0.55% Cu to the ore reserves, at a stripping ratio (that varies with the changes in cut-off) of 1.5 : 1, and gives the best alternative : total life of 28 years, net present value of $7.3 million, and a discounted cash flow rate of return of 8.8%.

COMPARATIVE ECONOMICS—FIXED TONNAGE PRODUCTION

Cutoff	0.6% Cu Tons	Avg. % Cu	0.5% Cu Tons	Avg. % Cu	0.4% Cu Tons	Avg. % Cu
Ore	100,000,000	0.90	100,000,000 40,000,000	0.90 0.55	140,000,000 55,000,000	0.80 0.45
Total ore	100,000,000	0.90	140,000,000	0.80	195,000,000	0.70
Tons waste	250,000,000		210,000,000		155,000,000	
Stripping ratio	2.5:1		1.5:1		0.8:1	
Annual production:						
Ore	5,000,000 tons		5,000,000 tons		5,000,000 tons	
Copper	80,000,000 lb		70,500,000 lb		61,000,000 lb	
Tons per day:						
Ore	16,000		16,000		16,000	
Waste	40,000		24,000		13,000	
Total	56,000		40,000		29,000	
Plant:						
Mine	$28,000,000		$20,000,000		$15,000,000	
Mill	24,000,000		24,000,000		24,000,000	
	52,000,000		44,000,000		39,000,000	
Life	20 yrs		28 yrs		39 yrs	
Costs:	Per ton	Per lb Cu	Per ton	Per lb Cu	Per ton	Per lb Cu
Mining	$0.45		$0.45		$0.45	
Stripping	1.00		.64		.36	
Milling & gen'l.	1.25		1.25		1.25	
	2.70	16.9¢	2.34	16.6¢	2.06	16.9¢
Treatment, etc.		5.9		6.0		6.2
Total direct cost		22.8¢		22.6¢		23.1¢
Profit @ 30¢ Cu:						
Per lb		7.2¢		7.4¢		6.9¢
Annual income	$ 5,760,000		$ 5,217,000		$ 4,209,000	
Amortization	2,600,000		1,571,000		1,000,000	
	3,160,000		3,646,000		3,209,000	
Depletion	1,580,000		1,823,000		1,601,000	
Taxable income	1,580,000		1,823,000		1,605,000	
Tax @ 52%	822,000		948,000		835,000	
Annual cash flow	4,938,000		4,269,000		3,374,000	
Total cash flow	98,760,000		119,532,000		131,586,000	
Payout	10.5 yrs		10.3 yrs		11.6 yrs	
Present worth @ 6% & 3%:						
Factor	10.29		12.01		13.54	
Amount	50,812,000		51,271,000		45,683,000	
Less plt. capital	52,000,000		44,000,000		39,000,000	
Net present worth	[1,188,000]		7,271,000		6,683,000	

. Comparative economics of grade cut-off related to fixed tonnage production rate.

* AFTER PLANT INVESTMENT

FIG.4.28 (Soderberg & Rausch,1968)

Cut-off:	0.60% Cu	0.50% Cu	0.40% Cu
Capital investment	$52,000,000	$44,000,000	$39,000,000
Annual cash flow	4,938,000	4,269,000	3,374,000
Payout	10.5 Years	10.3 Years	11.6 Years
Life	20 Years	28 Years	39 Years
Discounted cash flow return	7.1%	8.8%	8.2%

The table shown in Fig. 4.29 contains some generalized costing relationships for an open pit (after Mackenzie, 1982) that could be useful in assessing the economics of open pit projects.

GENERALIZED COSTING RELATIONSHIPS FOR AN OPEN PIT.
(After Mackenzie,1982)

FIG.4.29

CONSTRAINTS: In 1980 Canadian dollars.
M= mine capacity (tons of ore+waste per year)
$200\ 000 < M < 60\ 000\ 000$
T= mine and/or mill capacity (tons of ore per year)
$50\ 000 < T < 30\ 000\ 000$

Capacity(T)-Reserve(R) relationship: $T=5.63(R)^{0.756}$

Pre-production stripping cost for an open pit:
Unconsolidated overburden: $2.30 per cu.metre
Waste rock: $3.00 per cu.metre
Open pit operating cost (UOC=unit operating cost,$ per ton mined or milled):
$UOC= 378(M)^{-0.3691}$
Open pit equipment and maintenance facilities (UCC=unit capital cost,$ per annual ton of capacity):
$UCC= 304(M)^{-0.3255}$
Open pit sustaining capital cost (ACC=annual capital cost,$ per year):
$ACC= 0.1623(M)+81300$
Mill capital cost (UCC=unit capital cost,$ per annual ton of capacity):
Copper mill: $UCC= 18051(T)^{-0.4794}$
Copper-molybdenum mill: $UCC= 31520(T)^{-0.5114}$
Copper-zinc or lead-zinc mill: $UCC= 109021(T)^{-0.5903}$
Mill operating cost (UOC=unit operating cost,$ per ton mined or milled):
Copper mill: $UOC= 345.5(T)^{-0.3251}$
Copper-molybdenum mill: $UOC= 487(T)^{-0.3400}$
Copper-zinc or lead-zinc mill: $UOC= 1530(T)^{-0.4122}$
Mill sustaining capital cost (ACC=annual capital cost,$ per year):
$ACC= 0.01(UCC\ mill)(T)$
Working capital:
3 months operating costs (mine+mill+overhead)
Administrative and overhead operating costs:
20% of UOC (mine+mill)
Pre-production period: 2 years
Manpower for open pit and mill (N= number of persons employed):
$N= 0.02(M)^{0.5718}+0.074(T)^{0.5000}$
Accomodation (per person employed when necessary):
Houses= $70 000
Permanent bunkhouse facilities= $15 000
Amenities (per person employed when necessary)
Townsite= $20 000
Camp= $5 000
Road construction: $80 000 per km
Power facilities:
Peak load (kilowatts,KW): $KW= 7.9(T)^{0.4977}$
Isolated mines (assume diesel generating plant and low voltage): $capital\ cost= 6100(KW)^{0.8}$
Where utility power is available (assume transmission line,utility substation,and low voltage distribution):
For T< 500 000, use 27.6 or 41.6 KW line:
$capital\ cost= 1100(KW)^{0.8}+35000(D)$
For T> 500 000, use 110 KW line:
$capital\ cost= 1100(KW)^{0.8}+75000(D)$
where D= transmission distance,km

(Generalized costing relationships for an open pit. Continuation)

Mine recovery: MRF= 100%
Dilution: DF= 0%
Mill recovery factor:
        Copper mill: Cu-95%, Au-85%, Ag-85%
        Copper-molybdenum mill: Cu-90%, Mo-80%, Au-80%, Ag-80%
        Copper-zinc or lead-zinc mill: Cu-90%, Pb-90%, Zn-80%
                                       Au-60%, Ag-75%, Cd-75%
Concentrate grade:
        Cu-25%
        Mo-90%MoS$_2$
        Zn-55%
        Pb-60%
Concentrate transportation cost:
        Road: 3.0 cents per tonxkm
        Rail: <500 km, 5.0 cents per tonxkm
              500-1500 km, 3.0 cents per tonxkm
              >1500 km, 2.5 cents per tonxkm
        Transfer: $4 per ton concentrate at each transfer point

4.5. Optimal depth of an open pit

Due to the restrictions imposed by the maximum stable pit-slope angles, the increasing amount of waste to be removed as mining goes deeper will eventually limit the depth at which the open pit can be mined economically. The cost of transport also increases with depth, and it contributes to fix that economic depth limit.

When trying to determine the optimal depth of an open pit, it must be kept in mind that the alternatives not only include different pit depths, but also underground mining as well as different combinations of open pit and underground mine designs. Hence, alternative underground mining designs must be evaluated as well. The cost of the underground-mining alternatives will depend largely on the depth reached by the previous open pit stage, since it not only controls the hoisting costs but also, and more important perhaps, limits the tonnage available for underground mining; the revenues from underground mining must carry the cost of the necessary developments and very often the investment in a completely different selection of equipment.

A good example of determination of the optimal depth of an open pit is provided by Nilsson and Burgher (1982). Given an orebody, alternative pits reaching different depths are designed for a set of constraints. The pit design shown in Figs. 4.24 and 4.25 is one of these alternatives, and same as described for this pit layout, mining plans are estimated for the other alternative pits (assuming a plant capacity of 5 mill. tons per year). The alternative mining plans for various final pit depths are shown in Fig. 4.30.

FIG.4.30 .—Mining plans for various final pit depths (Nilsson & Burgher, 1982)

FIG.4.31 —Operating expenses for various open pit alternatives (Nilsson & Burgher, 1982)
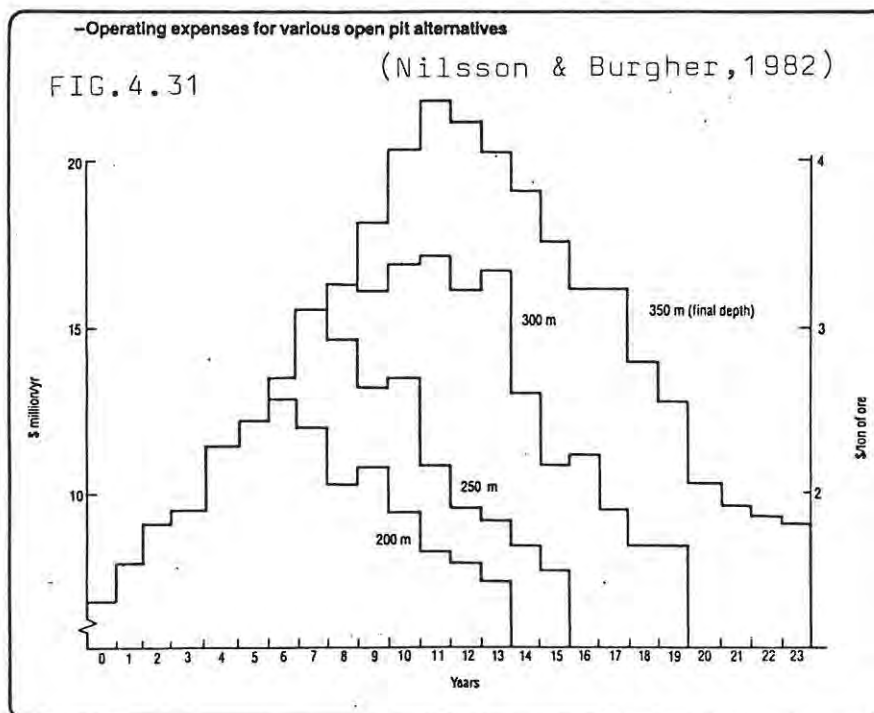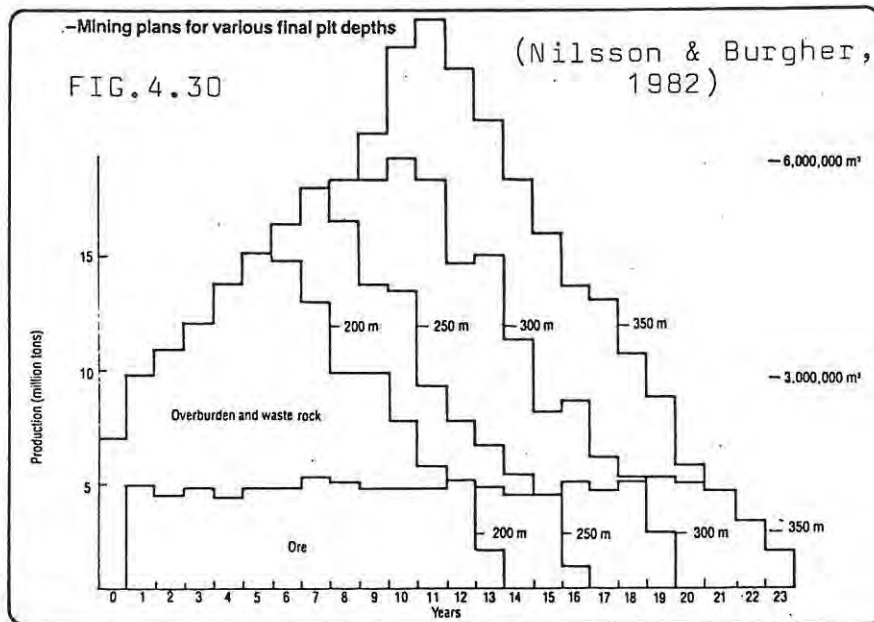
Then the operating costs (see Fig. 4.31), revenues and capital investments for each alternative are estimated based in the mining plans. Finally the Discounted Cash Flow Net Present Value of the different alternatives is calculated.

The Net Present Value of the underground mining of the whole orebody, and of alternative underground designs successively starting from the different pit bottoms downwards, have also been estimated.

In the tables below, the Net Present Capital Value (NPCV) for the different open pit and underground atternatives, and for combinations of both, has been recorded. It can be seen that at a price of $10 per ton of ore, open pit mining can proceed down to a depth of 300 m, but it would be more profitable to change over to underground mining at 250 m depth. With a price of $5 per ton of ore, the deposit can be mined with open pit methods down to a depth of 200 m, with future underground mining being unprofitable.

Final open pit depth (meters)	NPCV for open pit ($ million)		Final open pit depth (meters)	NPCV for underground mining at $10/ton ($ million)	
	$5/ton	$10/ton		When underground mining starts	When open-pit mining starts ✳
200	89	266	200	83	24
250	87	278	250	78	17
300	76	285	300	30	5
350	55	277	350	9	1

✳. Discounted at a 10% rate of interest.

Final open pit depth (meters)	NPCV for the deposit ($ million) ✳	
	$5/ton	$10/ton
200	89	290
250	87	295
300	76	290
350	55	278

✳. Open pit followed by underground mining.

The Net Present Capital Value (NPCV) at a rate of interest of 10% was used to evaluate the alternatives and was calculated as follows:

$$\sum_{n=1}^{n} \frac{I_n - O_n}{(1+i)^n} - C$$

where:

I_n = Annual income (constant)
O_n = Operating expense during a year
C = Capital investment
i = Interest rate, %
N = Lifetime for the open pit

4.6. Geological factors in open pit design

From the previous sections it is obvious that the size and shape of the orebodies, and the distribution of the ore within them, are the most important geological factors affecting the open pit design, particularly because of their relationship to the stripping ratio. Further discussion is presented in the section on grade control.

More specific problems are reviewed in this section.

Rock slope stability

Patton and Deere (1970) have found convenient to classify the several types of slope stability problems according to their magnitude in the three types shown in Fig. 4.32. This classification helps to discriminate those problems that affect the overall pit design from those affecting only localized portions of it or affecting only the operation.

The local slope failures (Type 1) are considered to extend a vertical distance of less than the height of one bench. They could be minimized by good slope design, but it is almost impossible to eliminate them completely without the use of excessively flat slopes or a costly slope-support system, such as gunite applied over wire mesh that is secured to rock anchors. It is unlikely that many mining operations could afford to eliminate all such failures, but experienced mine personnel could recognize the geological conditions leading to these failures at an early stage and adjust their mining operations accordingly. Such failures would not usually appreciably affect the overall mining operations, and the worst problem is that of the hazard to men and equipment.
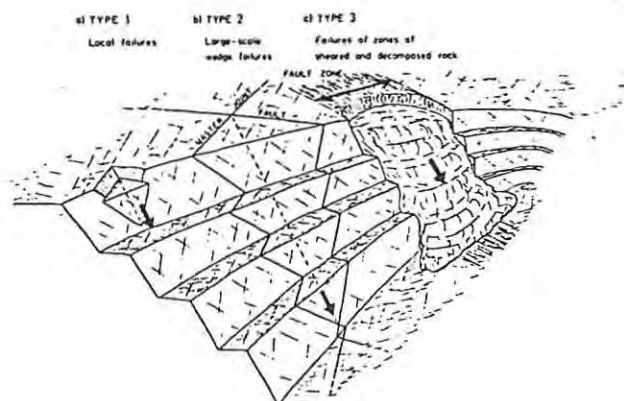
Three principal types of slope failures in open pit mines.

FIG.4.32   (Patton & Deere,1970)

The presence of two or more through-going discontinuities, such as a prominent bedding-plane joint or master joint combined with a fault (see Fig. 4.32 b) may produce a large scale wedge failure (Type 2), that could stop mining operations for months and change the economics of the entire project. Careful geological field work is required, since the geological conditions leading to the failure are often difficult to detect in advance; the structures may be separated by hundreds of meters at surface, and weathering could have obscured them. There may be little or no warning of imminent slope failure, until the line of intersection of the two geologic structures is exposed and failure takes place.

Failure may take place where the mine slopes encounter wide fault zones containing sheared and decomposed rock (Type 3 in Fig. 4.32), and the slope design used for the sound rock will have to be modified to account for the weaker material. Like Type 2, the slope may not fail until the excavation has exposed a large height of weaker materials.

Conditions leading to failures of Types 2 and 3 are important enough to be major considerations in the layout of an open pit mine, whereas the ones of Type 1 are normally dealt with during the operation of the mine. The paper by Martin and Piteau (1978) summarizes some criteria to select the correct berm width in final slopes to contain local failures (Type 1).

The distinguishing feature of slope stability problems in rock is that the failure planes conform closely to pre-existing planes of weakness, while the soil is considered to have relatively equal strength in all directions. In rock, the strength along a discontinuity may be only a small fraction of the strength of the intact material, and after large displacements the shear strength drops by several orders of magnitude and the residual strength becomes similar to that of soil with the same mineralogy (see Fig. 4.33). The low residual strengths obtained along rock surfaces which have undergone considerable displacement, is one of the reasons why uncemented faults and shears are so significant in slope stability problems. Fig. 4.34 illustrates one of those unfavourable situations; for an irregular joint

a) Shear Strength vs. Displacement   b) Shear Strength vs. Normal Stress
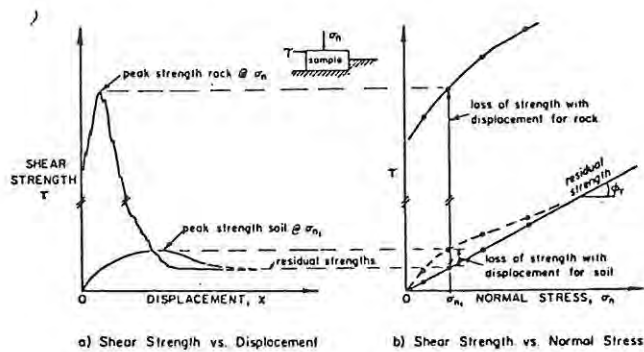
Comparison between soil and rock of the loss of strength with displacements.

FIG.4.33   (Patton & Deere,1970)

(a) the diagram shows that at small displacements the shearing
strength exceeds the shearing stresses, and the joint will remain
stable, but when the pit is excavated deeper to expose the uncemented
fault (c) at all displacements the shearing strength will not be
sufficient to resist the shearing stresses, and failure will result.

Irregularities associated with fault and shear zones can have
the effect of changing the equivalent angle of shearing resistance
of a rock surface by 15° or more, and they are important to record.
The role of such oriented irregularities along a fault surface is
shown in Fig. 4.35, in which two faults are in an unfavourable
orientation towards the slope.   The direction of movement in past
geologic history along fault No. 1 was north-south, resulting in
flutings in this direction, while the direction of movement along
fault No. 2 was east-west, resulting in east-west orientated flutings.
Fig. 4.35 b is a close-up sketch of a portion of the fault surfaces,
where τ_1 or τ_2 is the shear strength in the direction of fault
movements, striations, and deep fluting, and τ_3 or τ_4 the one
perperpendicular to that direction.   The mechanical significance of
the different orientations of the flutings is demonstrated in Fig. 4.35 c.
The fault plane No. 1 would fail, while fault plane No. 2 would not
slip downhill perpendicular to the flutings because the shear strength
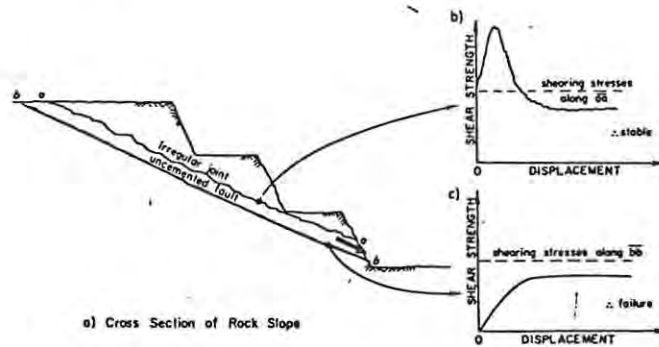
Significance of pre-existing displacements along faults.

FIG.4.34   (Patton & Deere, 1970)
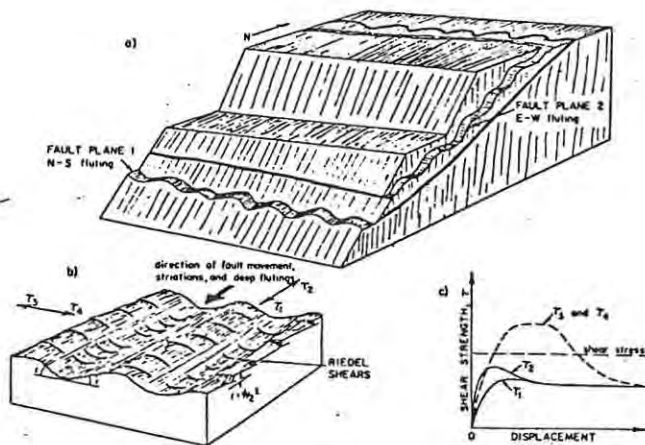
Details and effects of irregularities on fault surface.

FIG.4.35   (Patton & Deere, 1970)

τ_4 is greater than the shear stress.   The influence of irregularities of different orders of magnitude is discussed by Piteau (1970), and Patton and Deere (1970).

Some typical occurrences of shear zones in layered rocks are shown in Fig. 4.36.   Other important faults or shear zones include: (i) faults subparallel to or in secondary or conjugate alignment to regional faults, (ii) bedding plane faults in shales where they are

interbedded with other rock types, (iii) foliation shears in schists
and slates, and (iv) foliation shears along micaceous bands within
gneisses. The faults or shears mentioned in (ii) and (iii) are
particularly common in folded or inclined sediments adjacent to less
deformable rock (sandstone, basalt, quartzite or amphibolite).
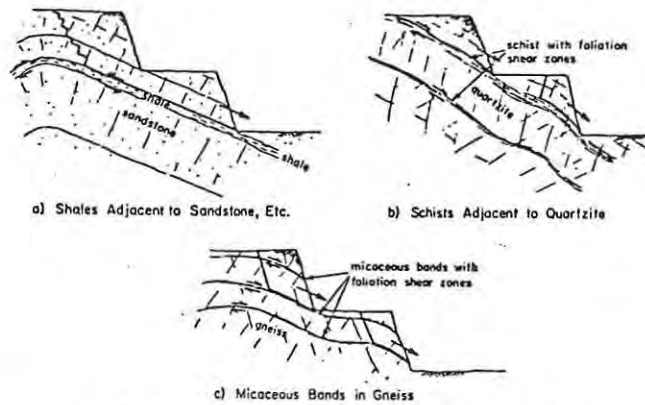
Typical occurrence of shear zones in layered rock.
FIG.4.36   (Patton & Deere,1970)

Fluid pressures within a rock mass act perpendicular to the sur-
faces of the discontinuities. When there are many joint sets with
many different orientations and when the joint spacing is small,
the fluid pressure within the rock mass can be treated in a similar
way to that used for soil slopes, but when the distribution of the
joints is anisotropic and the space between them is increased
unusual distributions of fluid pressure can result. In the example
in Fig. 4.37, two different stability situations can be originated
according to the water level and hydrostatic pressures as shown in
Fig. 4.38. The computed factors of safety indicate that all
combinations of blocks in case (a) are stable, but in case (b)
the block 1 might eventually fail after a series of forwards move-
ments (after each movement the fracture behind the block will greatly
increase and the water pressure will drop off until recharge takes
place).

Weathering on a rock mass greatly degrades its strength, changes its deformability and permeability characteristics, and develops a complex three-dimensional arrangement of residual soil, weathered rock and unweathered rock. In open pits the influence of weathering normally affects the upper benches, but in situations like the one shown in Fig. 4.39 it could affect the entire operation.

Intense argillic and sericitic hydrothermal alterations can produce effects similar to those of weathering. Large areas of weathered or hydrothermally altered rock may be found along wide fault zones, such as the one shown for the Type 3 slope failure in Fig. 4.32.
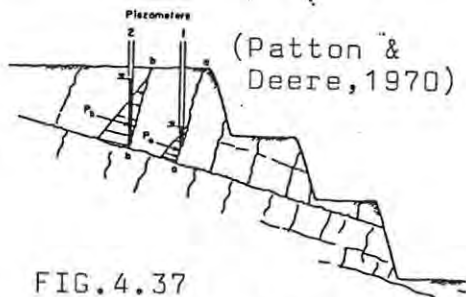
FIG.4.37 (Patton & Deere,1970)

Possible large differences in fluid pressures in adjacent rock joints.
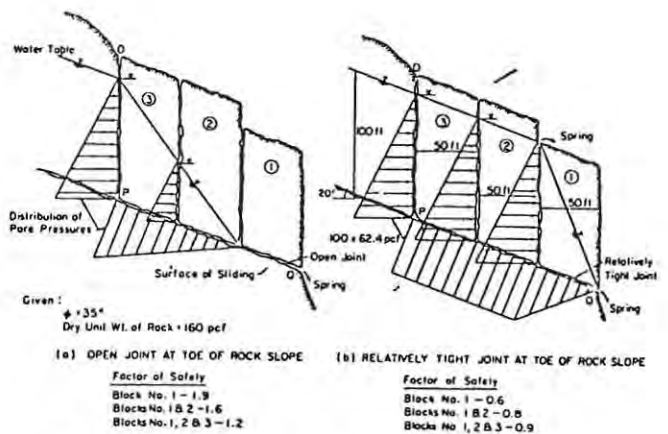
Effects of a geologic detail on pore pressures and stability.

FIG.4.38   (Patton & Deere,1970)
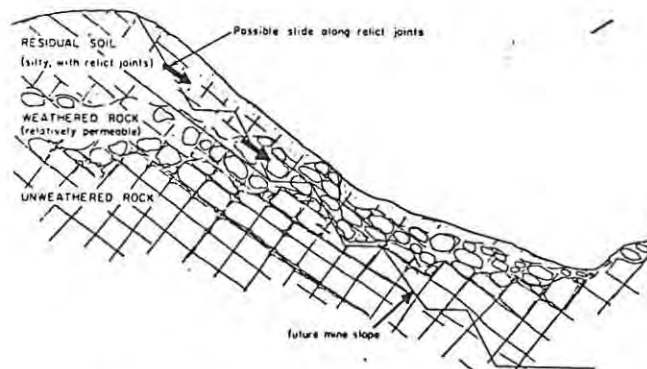
Mine slope affected by stability of residual soil and weathered rock.

FIG.4.39   (Patton & Deere,1970)

Other elements in rock slope analysis include shear strength along rock interfaces with soil and grout, influence of groundwater flow systems and the role of regional stresses, and they are discussed by Patton and Deere (1970).

The main models of failure in rock slopes are the circular, the failure along a discontinuity dipping towards the excavation, over-turning in vertically jointed rock and wedge failure in jointed or faulted rock.

The circular failure is characteristic of soils, homogeneous soft rocks, closely-spaced ramdomly jointed rock (wide faults or fracture zones, weathered or hydrothermally altered rock), and horizontally-bedded intact soft rock. The design chart for soft rock slopes with circular failure shown in Fig. 4.40, and the squares P, O and L in Fig. 4.41, taken from Hoek (1970, 1971), give an idea of the critical slope height - slope angle relationship in these materials. This relationship, as seen in square L of Fig. 4.41, shows that even for the very soft horizontally-bedded limestone considered, the critical slope heights are in excess of most open cast mine slopes currently under consideration. This confirms that for slopes in materials which would normally be classed as rock, failure of the intact material is unlikely and that instability would be controlled by structural discontinuities in the rock mass.
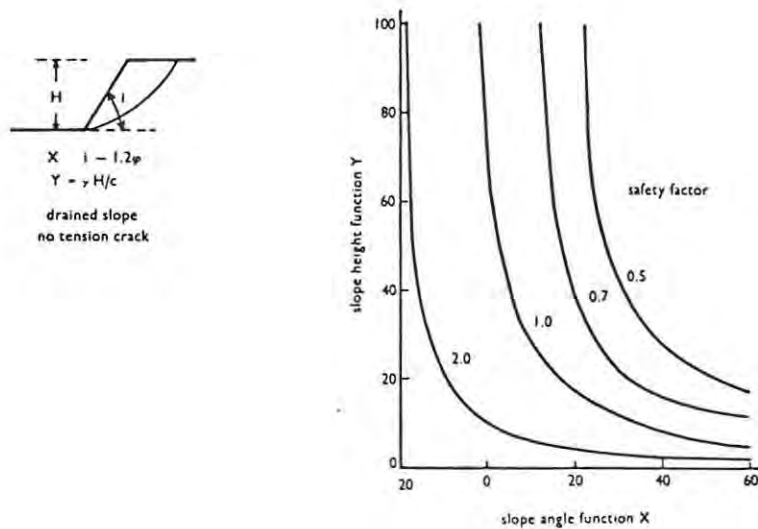
Design chart for soft rock slopes with circular failure (Hoek[3])

FIG.4.40   (Hoek,1970)
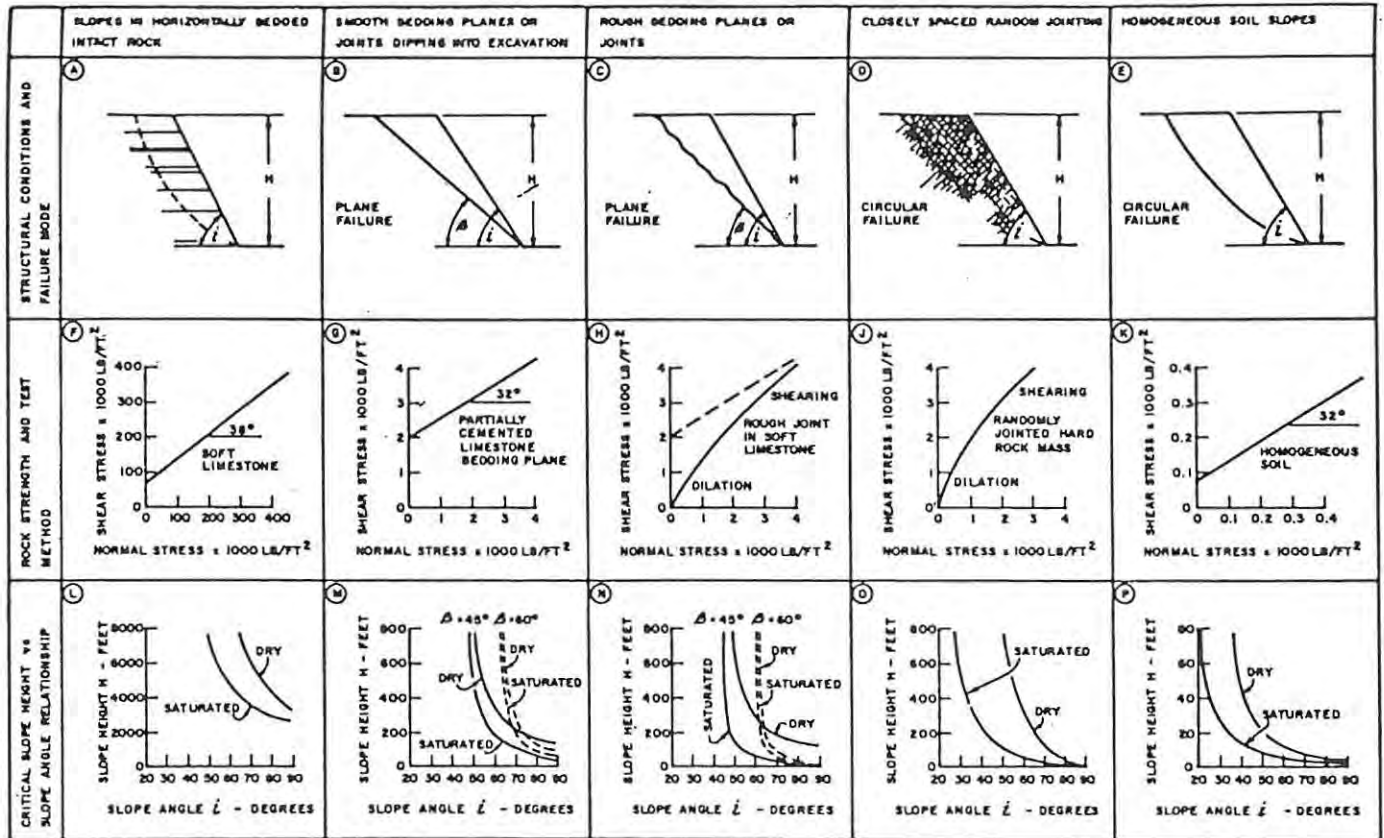
FIG.4.41 Influence of rock strength and structure upon slope stability.
(Hoek,1971)

Smooth bedding planes or joints dipping into the excavation remain very stable at low dip angles, but when the dip is greater than the corresponding angle of friction failure of the slope occur as a result of sliding on this plane.   This relationship can be seen in Fig. 4.42, for an intact soft limestone in which the bedding plane as a cohesion of 2000 lb/ft^2 and a friction angle of 32°.   It can be noted in this example that the most dangerous bedding plane is the one dipping at 61°. As the dip gets steeper than 61° the critical slope height increases again; the wedge of material above the failure plane is held up almost entirely by friction.   When the angle becomes close to 90° it is easy for the wedge to fail by overturning, especially when there is high groundwater pressure.   When the bedding planes are flat-dipping, the only possible mechanism of failure is the circular model and the slope is very stable.

INTACT LIMESTONE: UNIAXIAL COMPRESSIVE STRENGTH
σ_c = 288,000 LB./FT.^{2}
BEDDING PLANE: COHESION 2000 LB./FT^2 AND
FRICTION ANGLE = 32°

FIG.4.42   (Hoek,1971)

a.- TENSION CRACK IN A SLOPE.

b.- CRITICAL SLOPE CURVE FOR DRY AND WATER FILLED TENSION CRACKS AND NO TENSION CRACK.
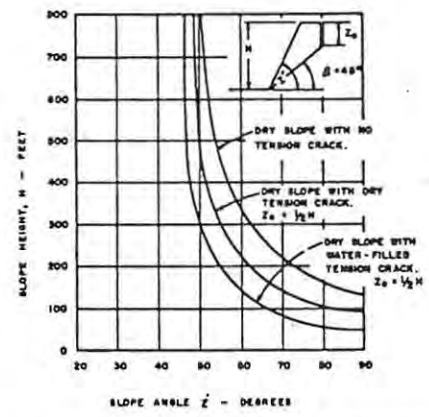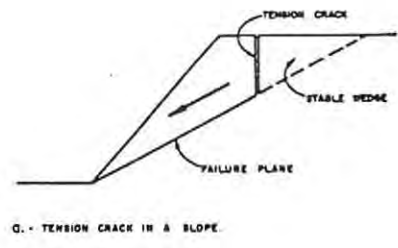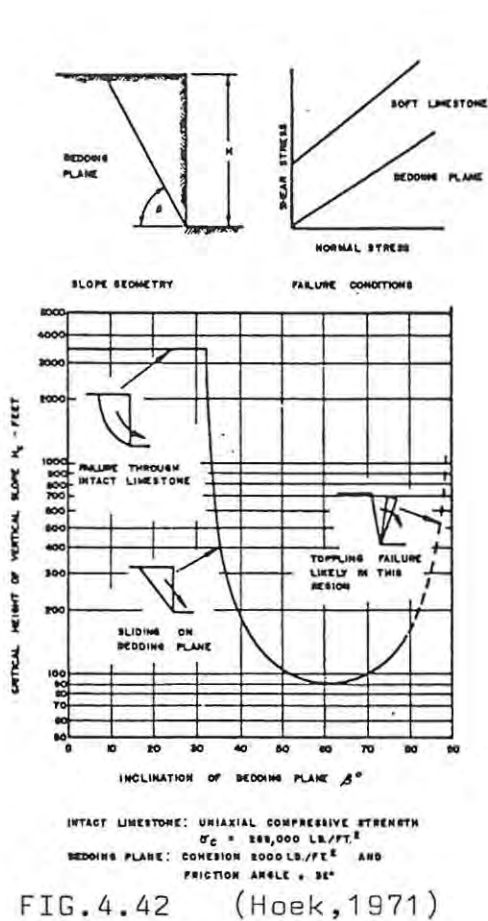
FIG.4.43   (Hoek,1971)

—Influence of tension cracks on slope stability.

The effects of irregularities along the rock discontinuities dipping into the excavation were previously discussed, and examples can be seen in squares M and N of Fig. 4.41.

The presence of a vertical or steeply dipping structural feature in the crest of the slope, or the development of a tension fracture in that position, can have a significant influence upon the stability of the slope (as shown in Fig. 4.43; note the effect of the presence of water in the fracture).

Fig. 4.44 illustrates a typical failure by overturning along nearly vertical bedding-plane faults. More than 40 large rock slides took place in 1968 in this section of the new Marginal Highway in Eastern Peru (Patton and Deere, 1970). The bedrock consisted of
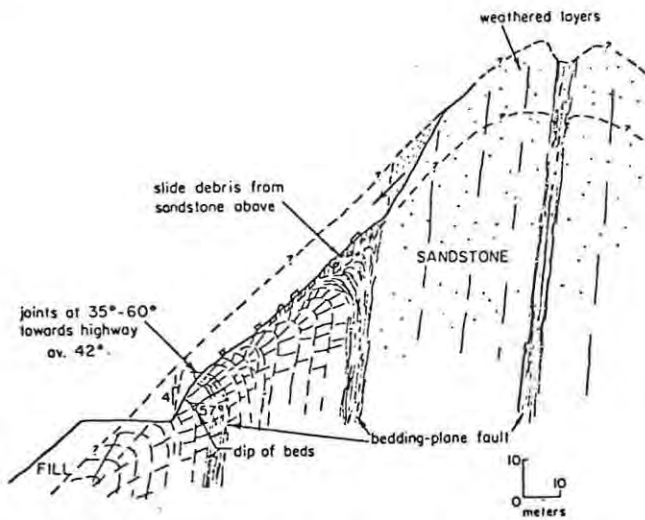
Typical section of slide—Marginal Highway, Peru.

FIG.4.44   (Patton & Deere,1970)
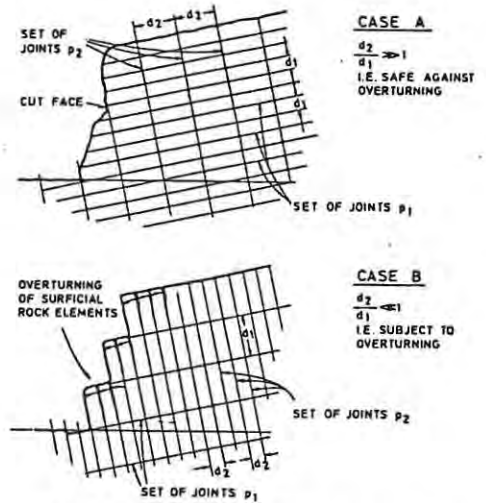
Joint spacing governing secondary failure of slope surface by overturning of rock elements

FIG.4.45   (John,1970)

interbedded sandstones and shales that had been steeply inclined.
The bedding-plane faults were confined to the weaker shale layers.
Following the erosion of the adjacent valley, mass movement or creep
of the surface layers of rock (overturning) produced a set of joints
dipping at low angles towards the valley.   During the building of the
road, rock slides took place along these joints, helped by the presence
of the near vertical bedding-plane faults in a way similar to the one
shown in Fig. 4.43.

Overturning can be controlled by the joint spacing, and the
simple example in Fig. 4.45 introduces the ratio d_2/d_1 (John, 1970).
Subvertical slabs with $d_2/d_1 \ll 1,0$ tend to produce "unsafe" slope
surfaces, particularly at overall slopes steeper than about 60°.

Fig. 4.46 shows some typical wedges in rock slopes.   The examples
at the top of the figure (1 to 6) can fail if the shearing strength is
overcome.   Fig. 4.47 shows how the wedge failure pattern can be
modified in the presence of intensely jointed rock.

Since the geometry on an opencast excavation is normally defined by the shape of the orebody being mined, the designer is not free to choose the most favourable pit orientation with respect to the regional structural pattern in the rock mass. The variation of the orientation of important structural features with respect to the slope face of the excavation should be kept in mind when choosing the pit slope angle; it may be found that slopes in certain portions of the pit can be appreciably steeper than the slopes in those portions in which the structural features are unfavourably orientated with respect to the slope face.

Structural mapping, in which the information is obtained from drilling and from surface exposures, and the determination of the shear strength of structural discontinuities are main tools in slope failure analysis. The derivation of the formulas to estimate the Factors of Safety for different models of failure can be found in Hoek (1970), Pentz (1971), Jennings (1970) and John (1970).
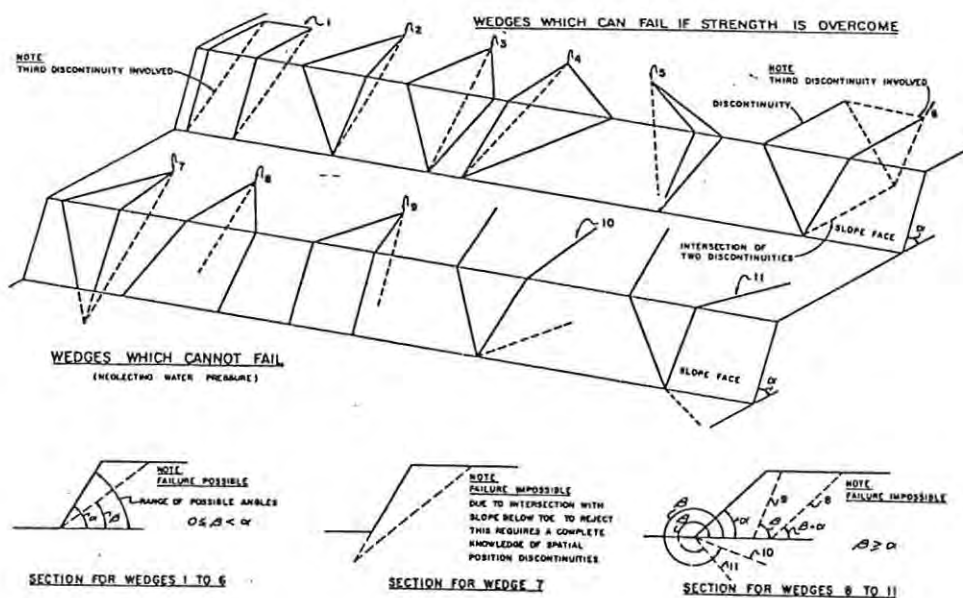
FIG. 4.46   (Pentz, 1971)
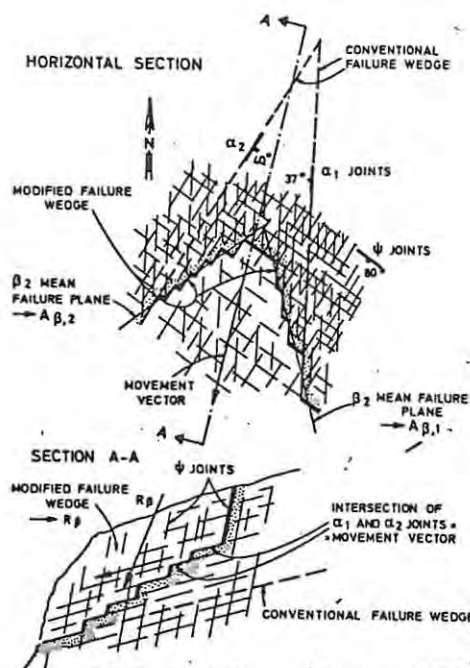
FIG.4.47   (John,1970)

. Failure pattern of modified wedge bounded by two mean failure
planes

Blasting

The reason to consider blasting in this section is that geology affects its results and, on the other hand, a properly planned blasting technique and sequence can be used for grade control purposes. Hence, a very rough understanding of blasting is necessary to plan grade control.

In general, the round design must provide an adequate tonnage of material, properly placed and with suitable fragmentation to ensure that loading, haulage, and subsequent disposal or processing are accomplished at the lowest possible cost. To that, from a grade control point of view, we could add that the blasting should limit, as much as possible, the mixing of broken waste and ore, and the spread of flyrock, thus reducing dilution and increasing ore recovery.

The bench geometry is one of the primary factors controlling the design of the blast. The bench height (L) is relatively constant and set by the working specifications of the loading equipment. This is

illustrated in Fig. 4.48 for a shovel; assuming there is no deliberate
heaping or scattering desired, the bench height can be approximated
as:

$$L = Lm \, Sf^X \quad (in \, ft.)$$

where Lm is the maximum cutting height of the shovel, Sf the swell
factor (ratio of the volume of a unit of weight of solid material
to that when broken), and x = 1/2 for box cuts and x = 1/3 for corner
cuts.
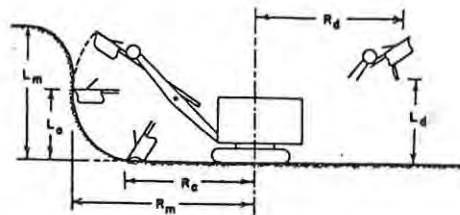
L_d= MAXIMUM DUMPING HEIGHT
L_m= MAXIMUM CUTTING HEIGHT
L_o= OPTIMUM HEIGHT OF CUT
R_c= RADIUS OF CLEAN-UP AT FLOOR
R_d= MAXIMUM DUMPING RADIUS
R_m= MAXIMUM CUTTING RADIUS

General working specifications of power shovels.

FIG.4.48   (Ash,1968)
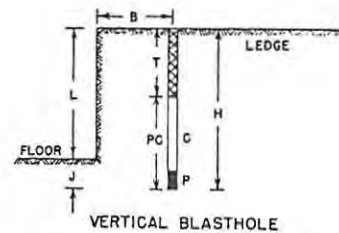
VERTICAL BLASTHOLE
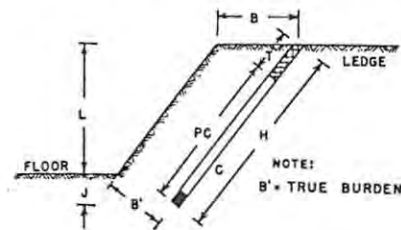
INCLINED BLASTHOLE

B = BURDEN,   C = MAIN CHARGE,   P = PRIMER,
T = STEMMING,   L = BENCH HEIGHT,   J = SUBDRILL,
PC = CHARGE LENGTH,   H = HOLE DEPTH.

Blasthole geometry.

FIG.4.49   (Ash,1968)

In a continuous cylindrical blast, after the shock wave has
passed, the gases begin widening the cracks made by the shock wave and
crushing the rock in the vicinity of the charge. In addition to crack
propagation, the gas effect provides the major source of movement of
the broken ore. Rocks with many planes of weakness require little
displacement to effect good shovel loading.

The rocks can be classified into two main categories: elastic and plastic-acting. Elastic rock should be thought of as a rock which can transmit a shock wave and is high in compressive strength, such as granite or quartzite; it makes good use of the shock energy from the explosive, forming cracks for the gas effect to work on. Plastic-acting rocks are relatively low in compressive strength and absorb shock energy at a much faster rate; rocks of this type are generally softer materials such as some limestones, sandstones and hydrothermally-altered porphyries, and in them the shock-energy part of the explosive reaction is wasted in plastic deformation, leaving most of the work to the gas effect. Because the ratio of gas effect to shock energy varies in different explosives, it is easy to understand why some explosives perform well in elastic rock and poorly in plastic-acting rock and vice versa. Some of the most difficult situations arise when mixtures of plastic-acting and elastic rock are encountered.

The competency, structural features, characteristic velocity of energy propagation, and moisture condition of a material influence the general pattern of the blast rounds.

Fractures serve as natural channels for the escape of explosive energy, and blasting must provide maximum confinement for longer periods of time. Fractures also absorb and scatter stresses propagated through a material, resulting in uneven breakage.

Jointing, bedding planes, schistosity and faulting strongly affect the results of blasting. Shear and tensile fracturing are comparatively easy to produce in the directions of jointing or bedding, and it is considered good practice to align rows of charges parallel with those directions (see Fig. 4.50).

For materials that have jointing planes of uniform strength and with nearly equal included angles, such as igneous rocks, the blasting patterns are relatively simple. However, in sedimentary rocks that usually have one particular direction along which jointing is most
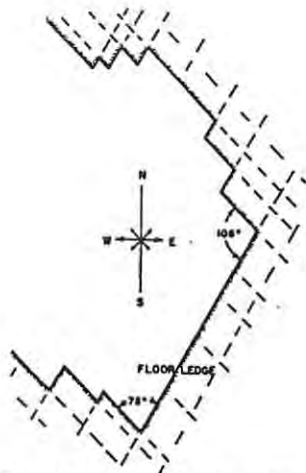
FIG.4.50 (Ash,1968)
Representative sedimentary vertical jointing system.

BOX CUT   CORNER CUT (Ash,1968)
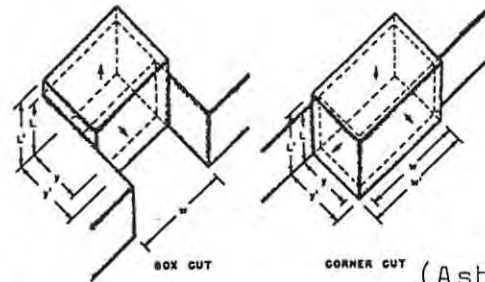
FIG.4.51   Swell from blasting before collapse.

pronounced and different angles between planes, the balancing of stresses is more difficult to attain. In the "tight corners" (75° angles) in Fig. 4.50 there is a tendency to overbreak, opening cracks along the jointing planes into the solid and resulting in coarse fragmentation in subsequent blasts. The tight corners also aggravate ground vibrations, frequently producing both air blast and flyrock. To reduce the adverse effects, blasts should be directed out of the open angles whenever possible, and spacing between adjacent blastholes should be extended along the jointing directions (if deliberate shearing is not intended).

All materials expand when broken from the solid with a characteristic swell factor (Sf), affecting the choice of an initiation-timing system and limiting the number of blasthole rows that should be used. For the box cut expansion will occur in two directions, while for the corner or side cut it will be in three directions (Fig. 4.51). If swell is assumed to be uniform, broken material should expand towards an open face to an amount equal to the original solid dimension divided by either the square root (box cut) or the cube root (corner cut) of the Sf ratio; the increase will vary from 5 to 33%, with 18% being considered average. The broken material will not stand vertically, and horizontal spread towards all open ends beyond that due to the swell will be a function of the natural angle of repose of the material,

of the moisture, and of the greater-than-normal throw from blasting (due to over-charging and effects of fractures channelling the scape of explosive energy). The implications of the broken ore spread pattern on grade control are obvious.

If the proper burden (see Fig. 4.49) has been selected for a single charge, the critical factors that control blasting result from a multiple-blasthole round would be the relative locations of all charges with respect to one another and the initiation-timing method employed.

Optimum depth of burden is controlled by the mechanism of cratering, illustrated for a single charge in Fig. 4.52. When a material is completely fragmented but not displaced, it is assumed that the critical burden has been attained; however, an amount less than the critical value is preferred since displacement is generally required for efficient loading. A simplified formula to estimate the burden is given by Ash (1968):

$$B = Kb \ De/12 \quad \text{(in ft)}$$

where De is the charge diameter in inches and the burden ratio, Kb, varies from 14 to 49 with an average value near 30 (reasonable for materials with an SG = 2.6 and an explosive having properties similar to 60% ammonia dynamite: SG = 1.3 and V_e = 12 000 fps). the formula

$$J = 0.3 \ B \quad \text{(in ft)}$$

gives an approximation of the minimum subdrilling.

In multiple blasthole design, for a bench height L and a cut depth y, the minimum spread Z of the broken material can be estimated as follows:

$$Z = \frac{1}{2S_f^x} \ (\ 2y \ (1 - S_f^x) + L) \quad \text{(in ft)}$$

where x = 1/2 for box cuts and x = 1/3 for corner blasts.

The holes should be so placed and detonated in such a sequence
that the rock breaks successively towards the free face. The ground
to be excavated should not be wedge shape as this requires larger
explosive density. Inclined blast holes provide uniform burden on the
first row of holes, help fragmentation by allowing the shock wave to
propagate further in inclined strata or jointing (see Fig. 4.53), and
decrease the possibility of stumps and incomplete tearing at the bench
toe. Nevertheless, with inclined blastholes the drilling cost is
higher due to the increased wear of the bit and the reduced penetration
rate.
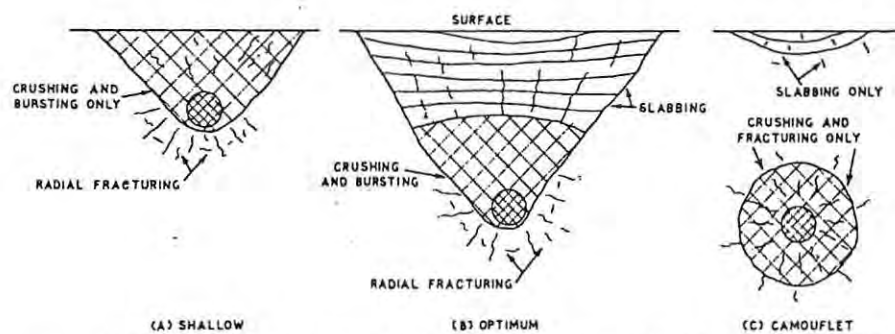
**Mechanism of cratering at (a) shallow; (b) optimum; (c) camouflet
depth of burial**
FIG.4.52   (Singh & Saluja,1972)

Some typical multiple blasthole designs (chevron, parallel and
diagonal) are shown in Fig. 4.54, in which the relationships between
the spacing of charges (S) and the burdens are given, and the direction
of the throw indicated.

4.7. Evaluation and grade control

When the geological factors controlling the distribution and grade
of the ore are well known, and allow the delineation of ore zones of
homogeneous characteristics by an economic grid of diamond drilling,
the cross-sectional method of ore reserve estimation is frequently the
one giving the best results. It gives not only average grades and
tonnages but also the distribution of the mineralization.
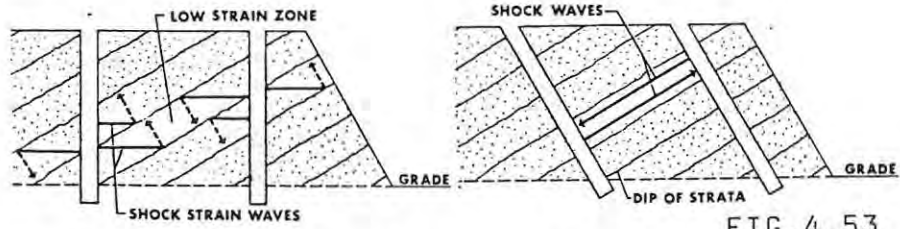
Making better use of shock-strain by allowing it to propagate

Angle holes drilled normal to bedding planes (Grant,1970)

FIG.4.53

DELAYED INITIATION FOR HOLES IN SAME ROW

SIMULTANEOUS INITIATION FOR HOLES IN SAME ROW

$S_1 = B_1$ $S_1 = 1.4B_2$ $S_2 = 2B_2$

Basic drill pattern relationship.

FIG.4.54
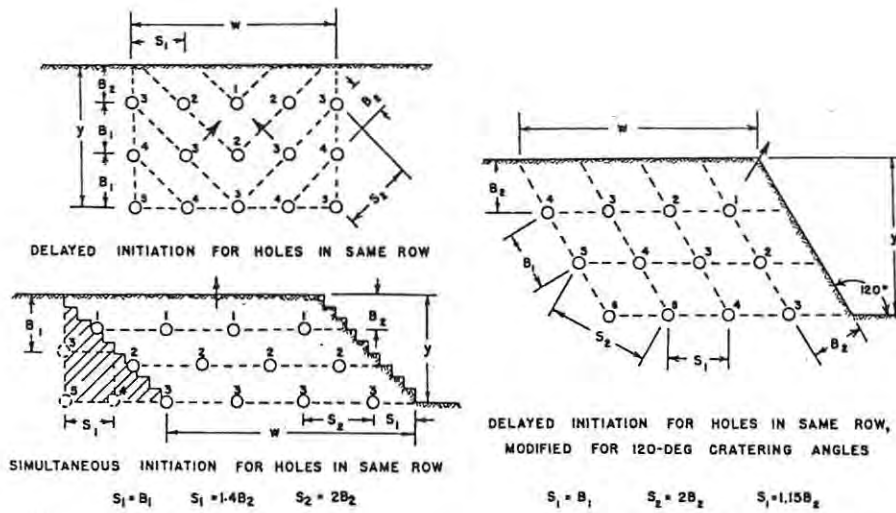
DELAYED INITIATION FOR HOLES IN SAME ROW, MODIFIED FOR 120-DEG CRATERING ANGLES

$S_1 = B_1$ $S_2 = 2B_2$ $S_1 = 1.15B_2$

Equilateral drill pattern for non-ideal cratering.

(Ash,1968)

Cross-sections perpendicular to the direction of maximum isotropy in the distribution of the mineralization are drawn, at a spacing controlled by the range of influence of the samples in that direction (as derived from the semivariograms). In each section mean grades are estimated by giving the samples areas of influence according to geological criteria; two adjacent samples on the same rock/ore unit are given lengths of influence up to the middle point of the line joining them, unless other geological criteria indicates a different length. Normal or lognormal means are estimated, according to the model followed by the distribution considered.

Geostatistical methods (kriging) have the advantage of giving confidence limits to the estimate, which helps in the process of risk analysis during the estimation of the economic feasibility of the project. How true these estimates are, or how much better than geological methods they are, is a debatable matter. An obvious advantage of geostatistical methods of ore reserve estimation is that in computer design of open pits a kriging program, followed by pit design and economic evaluation programs, can be applied to a common three-dimensional matrix containing the results of sampling and the basic geological data. Kriging also makes easy the short-term estimation of a mining block by blast-hole data on previously mined neighbouring blocks (as illustrated in Fig. 4.55 taken from Journal and Huijbregts, 1978, which seems to contain a mistake : the block being estimated, V, should be the middle block on bench B, and not on bench A).
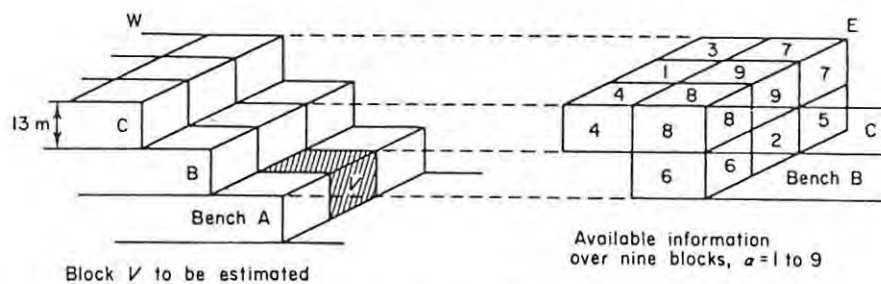
Short-term kriging plan at Chuquicamata.

FIG.4.55   (Journel & Huijbregts,1978)

The grade control practice aims to maximize the ore recovery during mining, minimize the dilution of that ore and the operating costs, and provide the concentrator with a constant feed of a product that follows the specifications used in the design of that plant (unless changes in policy require a mill feed conforming to different specifications).

The blasting technique is of great importance to reduce the dilution. At Rössing (Mining Magazine, 1978) both square and staggered drill patterns are used. The holes are tied up to allow parallel, diagonal or chevron pattern blasting on the first or second diagonal (see Fig. 4.54), and the blasting is planned to obtain the best grade control.

The geological mapping of the benches permits to delineate areas of ore and waste (it is complemented with the sampling of the blastholes). The blasting sequence is planned in such a way that the throw of the broken materials separates the ore from the waste. To achieve such separation the alternation of parallel and chevron patterns, parallel and diagonal, or chevron patterns with throws in different directions are frequently necessary. The blasting of wedges increases the explosive density and the drilling and blasting costs, and this factor must be taken into consideration; in general, it is worth to use these blasting techniques to separate from waste only ore of high grade or value.

For production purposes it is desirable to obtain maximum fragmentation of the rock, but this often results in excessive movement of the broken material, especially when more than four-row drill patterns are used, leading to intermixing between low and high-grade areas.

Blasting across the strike of the mineralized bodies results in a more even distribution of ore and waste available for loading than if blasting were done parallel to the strike. When the orebodies have a dip other than vertical or horizontal, the direction of the face advance also affects dilution. If, for example, mining proceeds at right angles to the strike of the orebodies and approaches one of them from its footwall side, then waste will still be mined at the toe when ore has already been encountered at the top of the face; in this

situation, perhaps a greater spread of the broken ore (particularly the one at the top of the bench) would be desired in order to separate it from the waste.

After the ore block has been blasted a visual evaluation of the broken material is often done, and the various zones of ore, low-grade material and waste, and zones with deleterious impurities or zones of different mineralogy, are demarcated using luminous tapes and signboards.

At the Scully iron and manganese ore mine in Canada (O'Leary, 1979), an integrated ore reserve calculation/blasting and shovel-trucks scheduling computer program is in use (see Fig. 4.56). The object of the grade control program is to provide a blend of ore, suitable for mill feed, at the crusher. This is achieved in two stages. Firstly the grade of ore in front of each shovel is calculated. As far as possible only one ore type should be included per shovel, and to this end limit stakes may be placed at the operating face to restrict the run of a shovel.

The second part of the program is designed to find a suitable schedule for shovel and truck combinations using the grade and tonnage computed in the first part of the program. Using the haulage distance from the crusher to a shovel and back, the maximum tonnage that one truck can haul in a shift is calculated. Then combinations of two, three and four shovels together are calculated varying the number of trucks serving each shovel. In theory it is possible to assign 5 trucks to one shovel, but in practice no more than 2 trucks are assigned and the combinations doubled and trebled to use all the available trucks. A number of restrictions are included in the program to limit the output of the computer. These are Fe and Mn contents, weight recovery of Fe, trucks per shovel (2) and shovels per combinations (4). The grading engineer selects those combinations which best meet the requirements for that day and inform the operating personnel. An example of the computer output is shown in Fig. 4.57.

Other examples of blending practice are found in Fraser (1971) and Steenkamp (1979).
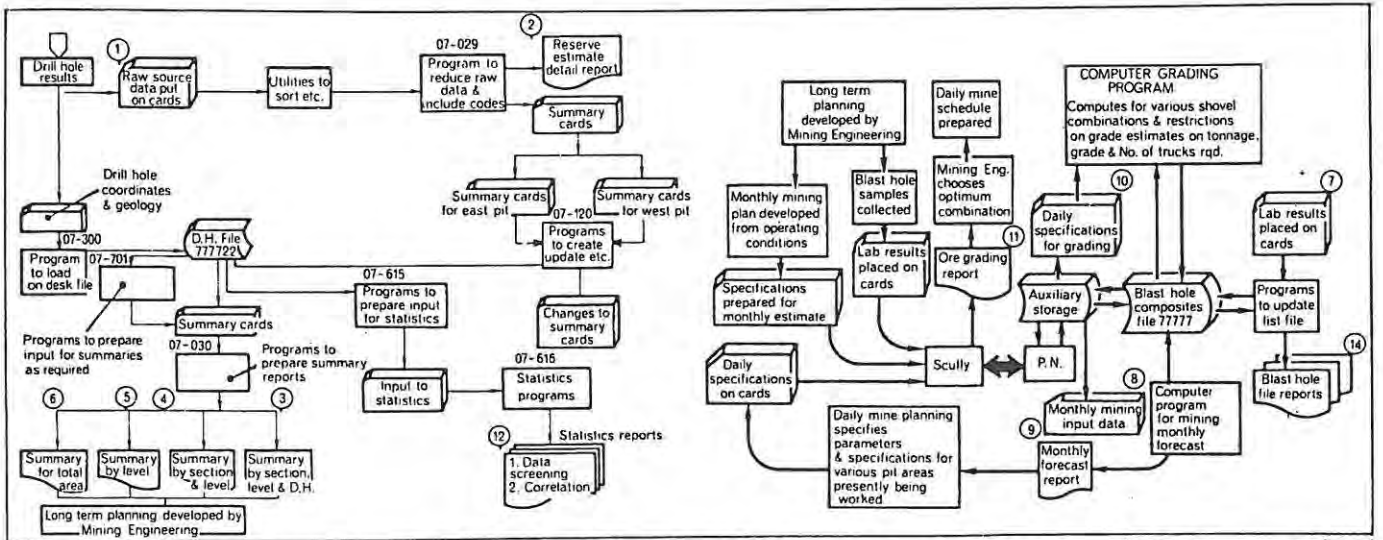
Left: Mining systems drill hole evaluation. Right: Blasthole evaluation for ore grading. FIG.4.56 (O'Leary,1979)

The obvious conclusion is that to estimate recoverable ore reserves in an open pit operation, the grade control program and its limitations must be considered. If orebodies, say, less than five meters wide cannot be separated effectively then their dilution must be taken into account. In the initial estimate of ore reserves a block, as a whole, could have been considered as waste, but perhaps good grade control practice could recover some tonnage of above cut-off ore from it. The grade control practice tend to produce sorting (and modifies those ore reserve estimates that have not taken it into consideration).

FIG.4.57

Example of machine output.

(O'Leary, 1979)

```
// JCB GRADE                                                    14.41.0
// EXEC GRADE

                              WABUSH MINES

                   GRADING COMBINATIONS FOR ALL SHOVELS
SHVL   WT REC    FE    SIL    MN    MAG  COARSE  DIST  MAX TONS   MAG

 1-U    40.27  63.62  5.85  1.67  4.07   0.0   7200. 4832.40  16.87
 3-0    56.23  63.90  6.70  0.98  0.0    0.0   9000. 6747.60   0.0
 4-0    39.82  66.86  3.42  0.0   1.16   0.0   5500. 4778.40   4.81
 5-0    32.62  64.83  3.16  2.07  3.17   0.0   5000. 3914.40  13.14
 6-0    54.16  65.12  3.39  1.30  0.01   0.0  10500. 6499.20   0.04
 7-0    38.75  65.71  2.78  1.28  4.18   0.0  11500. 4650.00  17.33
 7-W    28.43  66.16  2.94  1.23  2.82   0.0  11500. 3411.60  11.69
                                         RESTRICTIONS
HI-SIL-LO  HI-MN-LO  HI-HT REC-LO  HI-MAG-LO   HI-FE-LO  HI-COARSE-LO

 6.00   0.0   2.00  1.37 50.00 37.00 99.99  0.0   0.00 50.00 99.99  0.0

        TONS    WT   TONS   FE    SIL    MN    MAG
SHVL   CRUDE   REC   CONC  CONC   CONC  CONC   CONC  LOADS  TRUCKS

 1-0    2884.  40.27  1162.  63.62  5.85  1.67  16.87  52.4   2.00
 5-0    1761.  32.62   574.  64.83  3.16  2.07  13.14  32.0   1.00
 TOT.   4645.  37.38  1736.  64.03  4.96  1.81  15.64  84.5   3.00

 1-0    4327.  40.27  1742.  63.62  5.85  1.67  16.87  78.7   3.00
 5-0    1761.  32.62   574.  64.83  3.16  2.07  13.14  32.0   1.00
 TOT.   6088.  38.06  2317.  63.92  5.19  1.77  15.95 110.7   4.00

 1-0    1442.  40.27   581.  63.62  5.85  1.67  16.87  26.2   1.00
 6-0    1134.  54.16   614.  65.12  3.39  1.30   0.04  20.6   1.00
 TOT.   2577.  46.39  1195.  64.40  4.59  1.48   8.23  46.8   2.00

 1-0    1442.  40.27   581.  63.62  5.85  1.67  16.87  26.2   1.00
 6-0    2269.  54.16  1229.  65.12  3.39  1.30   0.04  41.2   2.00
 TOT.   3711.  48.77  1810.  64.64  4.18  1.42   5.45  67.5   3.00

 1-0    2884.  40.27  1162.  63.62  5.85  1.67  16.87  52.4   2.00
 6-0    1134.  54.16   614.  65.12  3.39  1.30   0.04  20.6   1.00
 TOT.   4019.  44.20  1776.  64.14  5.00  1.55  11.06  73.1   3.00

 1-0    2884.  40.27  1162.  63.62  5.85  1.67  16.87  52.4   2.00
 6-0    2269.  54.16  1229.  65.12  3.39  1.30   0.04  41.2   2.00
 TOT.   5153.  46.39  2390.  64.40  4.59  1.48   8.23  93.7   4.00

 1-0    4327.  40.27  1742.  63.62  5.85  1.67  16.87  78.7   3.00
 6-0    1134.  54.16   614.  65.12  3.39  1.30   0.04  20.6   1.00
 TOT.   5461.  43.16  2357.  64.02  5.21  1.58  12.49  99.3   4.00

 1-0    1442.  40.27   581.  63.62  5.85  1.67  16.87  26.2   1.00
 7-0    2131.  38.75   826.  65.71  2.78  1.28  17.33  38.7   2.00
 TOT.   3573.  39.37  1406.  64.85  4.05  1.45  17.15  65.0   3.00

 1-0    1442.  40.27   581.  63.62  5.85  1.67  16.87  26.2   1.00
 7-0    3156.  38.75  1239.  65.71  2.78  1.23  17.33  59.1   3.00
 TOT.   4638.  39.23  1819.  65.05  3.77  1.41  17.19  84.3   4.00
```

5. CHOOSING AN UNDERGROUND MINING METHOD

The basic objective when selecting an underground method to mine a particular ore deposit is maximizing the profits derived from the operations; however, other factors, such as optimizing the productivity, maximizing the extraction of the ore, and providing safe working conditions, must be considered in the selection as well.

Sometimes the characteristics of the orebodies and surrounding country rocks are quite distinctive, and they may dictate one particular method or the immediate exclusion of other methods. In other cases, the conditions may favour the application of several methods, which then must be compared and evaluated. Furthermore, very often the successful application of a mining method requires more than textbook knowledge, and a creative mind is essential in order to modify or adapt a mining method, or a combination of methods, to the local conditions; for this reason (and because of the dangers of "pigeonholing"), the guidelines given in this chapter are kept as simple and general as possible, and the details of the range of application of each mining method are discussed in the corresponding chapters.

Since a project started to-day will require several years to reach production, and it is expected to produce ore for many years after that, an up-to-date knowledge of the latest developments in mining techniques, and a feeling for the future trends, are necessary to design a successful project. It is advisable to incorporate features that allow flexibility and growth in the mining system; in particular, the dimensions of the shafts and accesses, and the ventilation systems should be designed allowing for flexibility and growth. At present, many mines no longer can accommodate the increasing dimensions of the new equipment, and the voluminous exhaust gases emitted by those machines.

The process of selecting a mining method depends upon the information available. It is rarely possible to do more than a preliminary study from the core drilling observations and other surface investigations, and its complement with additional drilling and information gathering from underground workings is frequently essential to produce a final mining plan. Even at this preliminary stage the planning and evaluation of the mining

method, together with a preliminary selection of equipment, require careful study and consideration; once the plans are set and the development begins it is extremely difficult and costly to change to an alternative method, and though it may be found necessary to modify details (say, in stope delineation), the basic principles of ore extraction should remain a part of the final layout.

Most of the factors that physically influence the choice of a mining method are geological in nature, such as the dip, shape and size of the orebody, its continuity, the nature of its contacts and the distribution of the values within it, and the rock mechanics of the orebody and of the surrounding country rock. The Table in Fig. 5.1 provides rough geological and rock-mechanical criteria for selecting large-scale mining methods.

Geological and Mechanical Criteria for Large-Scale Mining Methods

Method	Ore-Body Characteristics						Ore-Body Configuration							
	Ore Strength			Waste Strength			Beds		Veins		Mass	Ore Dip		
	Weak	Moder-ate	Strong	Weak	Moder-ate	Strong	Thick	Thin	Narrow	Wide		Flat	Moder-ate	Steep
Room and Pillar*		x	x		x	x	x	x				x	x	
Sublevel Stoping†		x	x			x			x	x	x			x
Shrinkage		x	x	x	x				x	x			x	x
Cut-and-Fill		x	x	x	x				x	x	x		x	x
Square Set	x			x	x				x	x	x		x	x
Block Caving‡	x	x		x	x					x	x			x
Sublevel Caving		x	x	x	x					x	x			x
Longwall	x	x		x				x				x	x	

Source: US Bureau of Mines, OFR 36-74 (1974).
* Uniform thickness and grade.
† Regular hanging and foot walls.
‡ Strong fractured rock also can be caved.

FIG.5.1   (Halls,1982)

Steep dips range from the angle controlling gravity flow, i.e. about 50°, to vertical. Flat dips are more difficult to define, since they are connected with equipment capabilities (the upper limit being the maximum gradient at which mobile equipment can operate); since mine workings can be oriented at an angle to the dip, manageable grades can be achieved for mobile equipment even in medium dips. The medium, or moderate, dips normally range from 20° to 50°.

The strength of a rock is a very subjective matter : what might be considered strong rock in a coal mine could be viewed quite differently in a hard-rock mining environment. In a preliminary stage, the strength of the rock can be inferred from the diamond drill core recoveries, from measuring the uniaxial compressive strength of core samples of different rock types, and from the analysis of the different geological discontinuities observed on surface and in the drill cores.

The delineation of the orebodies and the estimation of ore reserves are based on the definition of a cutoff grade, especially when the boundaries of the mineralization are not distinct. The determination of a cutoff grade (see section 2.1) assumes that the mine plan, schedule and costs are known, at least approximately, thus closing a vicious circle. Therefore, at this stage different cutoff grades can be assumed and orebodies of varying tonnages and average grades can be outlined. Each alternative may involve completely different mining systems. In the example in Fig. 5.2, outlining a small high-grade orebody, case A, makes necessary to plan a very modest size mining operation where the investment in infrastructure, development and equipment must be minimal (i.e. a labour-intensive mining method); in case B an average size and ore-value alternative has been outlined; and in case C the large low-grade orebody delineated will require large-scale mining, with high productivity and low operational costs (e.g. block caving mining).

Example of Grade and Ore Relations

Case	Cutoff Grade	Average Grade	Ore Reserves, mt
A	4.0%	5.5%	500 000
B	1.5%	2.5%	5 000 000
C	0.5%	0.9%	50 000 000

FIG.5.2　(Hamrin,1982)

As seen in the previous example, widely differing plans can be created for one particular mineralization, and the relationships between production capacity, ore grade, and available reserves are factors that must be included in the selection of a mining method. The selection of an alternative is based on the relative profitabilities of the different

mining plans thus created. The selected alternative is then studied in detail and an accurate cutoff grade is determined (hence fixing the final outline of the orebody).

In selecting a mining method, the anticipated cost of mining exerts a major influence (a cost comparison is shown in Fig. 5.3). However, the selection is not reduced to simply finding the least costly method; the characteristics and advantages of the different mining methods (e.g. selectivity, dilution, ore recovery, rate of production, etc.) must also be considered, and matched to the characteristics of the orebody. (Some generalized costing relationships in underground mining are shown in Fig. 5.4).

Cost Comparison of Mining Methods

Mining Methods	1977 Costs, $/net t ($ per net st)			Productivity, net t (net st) per manshift			Labor As % Total Cost
	Low	High	Average	Low	High	Average	
Room-and-Pillar Stoping	1.28 (1.16)	4.96 (4.50)	2.26 (2.05)	9.1 (10)	136.1 (150)	12.7 (14)	44
Cut-and-Fill Stoping	3.38 (3.07)	16.53 (15.00)	7.72 (7.00)	2.7 (3)	10.9 (12)	8.2 (9)	57
Sublevel Stoping	1.16 (1.05)	6.61 (6.00)	2.70 (2.45)	9.1 (10)	27.2 (30)	17.2 (19)	58
Sublevel Caving	4.41 (4.00)	8.82 (8.00)	5.51 (5.00)	10.9 (12)	25.4 (28)	12.7 (14)	64
Block Caving	0.83 (0.75)	1.60 (1.45)	1.60 (1.45)	9.1 (10)	45.4 (50)	13.6 (15)	55
Shrinkage Stoping	1.93 (1.75)	4.46 (4.05)	4.46 (4.05)	2.7 (3)	9.1 (10)	4.1 (4.5)	54

Adapted from: Gentry, D.W., 1976, "Procedure for Determining Economics of Small Underground Mines," *Mineral Industries Bulletin*, Vol. 19, January.

FIG.5.3   (Halls,1982)

The decrease in mining costs involves an increase in productivity that has become synonymous with mechanization, replacing manual labour with machines. Productive mechanization is closely related to obtaining high utilization of the equipment, which, in turn, is achieved by using a mining method with several working places within easy reach of each other (e.g. sublevel caving), or, alternatively, using a method in which a large volume of production is concentrated into a few places, allowing the machinery to be less mobile. In terms of mechanization, room-and-pillar and sublevel caving have a slight advantage over block caving and sublevel stoping. A common basis for comparing the efficiency of mining operations calculates the tons-per-manshift (see Fig. 5.3); this is the output from the mine per working shift, divided by the number of underground workers (also including the labour not directly involved in ore production).

FIG.5.4 (Mackenzie,1982)

SOME GENERALIZED COSTING RELATIONSHIPS FOR UNDERGROUND MINING.

AFTER B.MACKENZIE (1982)
(Monetary values expressed in constant 1980 Canadian dollars).

Definitions:
- T= mine or mill capacity,tons of ore per year.
- R= recoverable ore reserves,tons.
- UCC= unit capital cost, $ per annual ton of capacity.
- UOC= unit operating cost, $ per ton mined or milled.
- ACC= annual capital costs, $ per year.
- CC= capital cost, $.
- S= shaft depth, metres.
- W= average ore width, metres.
- E= stope width, metres.
- N= number of persons employed.

Capacity-reserve relationship:
Within the limits $50000 < T < 6000000$

$$T = 4.22(R)^{0.756}$$

rounding T,
- to the nearest 10000 for $50000 < T < 250000$
- to the nearest 25000 for $250000 < T < 500000$
- to the nearest 50000 for $500000 < T < 1000000$
- to the nearest 100000 for $1000000 < T < 6000000$

Shaft sinking:
Timber shaft for $50000 < T < 500000$

$$CC = (0.003389(T) + 3513)(S)$$

Concrete shaft for $500000 < T < 1500000$

$$CC = (1525(T)^{0.1204})(S)$$

Multiple shafts required for $T > 1500000$

Shaft equiping and surface installations:

$$CC = 7244(T)^{0.5136}$$

Pre-production underground mine development:

$$CC = 60(T)/(W)^{0.8}$$

Pre-production period:
- 2 years for $50000 < T < 100000$
- 3 years for $100000 < T < 500000$
- 4 years for $500000 < T < 1500000$
- 5 years for $T > 1500000$

Underground mine plant equipment:

$$UCC = 3000(T)^{-0.4095}$$

Underground mine sustaining capital cost:

$$ACC = 137.47(T)^{0.6791}$$

Underground mine operating cost:
Blasthole open stoping (sublevel stoping)

$$UOC = 2853(T)^{-0.3837}$$

Shrinkage stoping

$$UOC = 1774(T)^{-0.3000}$$

Cut-and-fill stoping

$$UOC = 1736(T)^{-0.3122}$$

Manpower for underground mine and mill:
Blasthole open stoping (sublevel stoping)

$$N = 0.73(T/E)^{0.5} + 0.074(T)^{0.5}$$

Shrinkage stoping
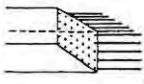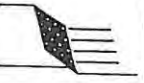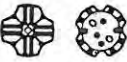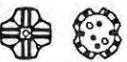
$$N = 0.053(T)^{0.7}/(E)^{0.5} + 0.074(T)^{0.5}$$

Cut-and-fill stoping
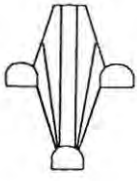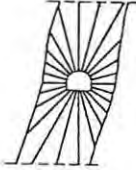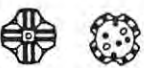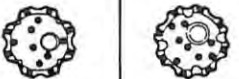
$$N = 0.064(T)^{0.7}/(E)^{0.5} + 0.074(T)^{0.5}$$

(Additional costing relationships,relevant to underground minig as well,
can be found in the section on economics of open pit mining).

Mining techniques have become totally dependent upon machines of various types. Selecting a machine for a given type of work, and matching its capacity to the required output have become two of the most important tasks in mine planning and evaluation. A very simple review of commonly used equipment units, their applications in mining methods, and their potential productivities are shown in Figs. 5.5, 5.6 and 5.7.

Mining Method	Room- and- Pillar Mining		Cut- and -Fill Stoping		Shrinkage Stoping	
Drilling and Blasting Technique	Drifting and slashing Frontal benching	Vertical or downward benching	Roof-drilling or overhand stoping	Frontal stoping, breasting	Overhand stoping	Frontal stoping, breasting
Applicable Drilling Equipment	Mechanized Drifting Jumbo	Mechanized Airtrack drill	Mechanized Light Wagon Drill	Mechanized Drifting Jumbo	Hand-held Stoper Drill	Hand-held Airleg Drill
Drilling Data						
Hole diameter, mm	38 – 48	64 – 76	33 – 38	38 – 48	29 – 33	29 – 33
Hole depth, m	3.0 – 5.5	as required	3.0 – 4.0	3.0 – 4.0	2.0 – 2.5	2.0 – 3.5
Drilling Equipment Performance:						
With pneumatic rock drill m/h	60 – 75	15 – 25	20 – 40	60 – 70	8 – 12	10 – 15
With hydraulic rock drill m/h	90 – 110	(25 – 35)	na	90 – 110	na	na
Drilling – blasting Production, m^3 rock per drilled metre	1.5 – 2.0	3.0 – 4.0	0.9 – 1.2	1.0 – 1.4	0.7 – 0.9	0.7 – 0.9

Common drilling methods, equipment, and performance.
FIG.5.5   (Hamrin,1982)

Mining Method	Sublevel Caving	Sublevel Stoping		Sublevel Stoping	
Drilling and Blasting Technique	Fan drilling	Ring drilling		Parallel drilling	
Applicable Drilling Equipment	Mechanized Fandrill	Bar-and-arm rigging	Mechanized Ring drill rig	Mechanized Airtrack with DTH hammer and high pressure	
Drilling data					
Hole diameter, mm	48 – 51 (64)	48 – 51	48 – 51 (64)	105 – 115	152 – 165
Maximum hole length, m	12 – 15	15 – 20	15 – 25	50 – 60	50 – 60
Drilling Equipment Performance:					
With pneumatic rock drill m/shift	200 – 240	50 – 60	100 – 120	50	50
With hydraulic rock drill m/shift	240 – 300	na	120 – 180	na	na
Drilling – Blasting Production, m³ rock per drilled metre	1.8 – 2.3	1.5 – 2.5	1.5 – 2.5	8 – 10	14 – 18

Common long-hole drilling methods, equipment, and performance.

FIG.5.6   (Hamrin,1982)
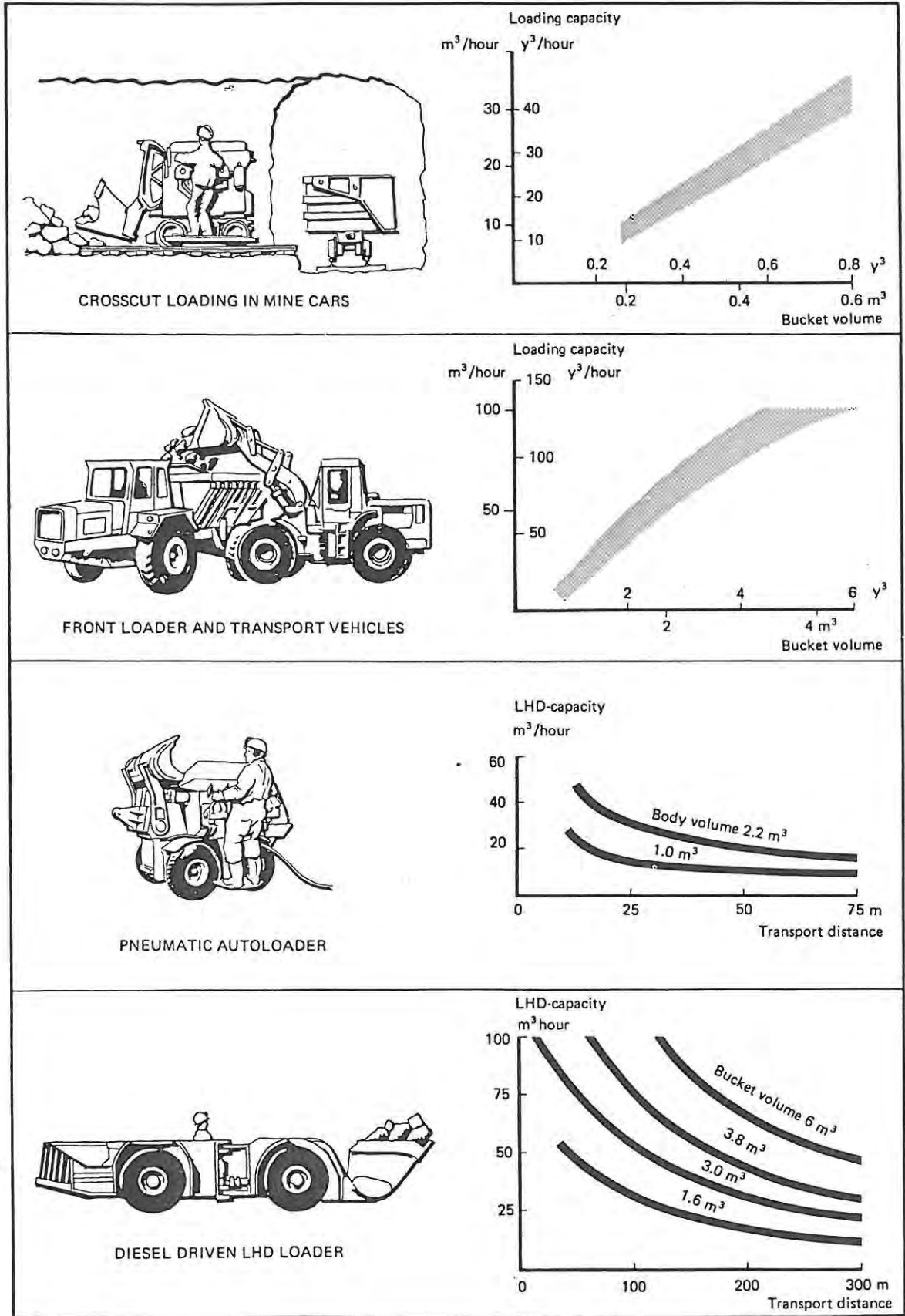
Common loading methods and performance.

FIG.5.7 (Hamrin,1982)

6. CAVING METHODS

6.1. Block caving

The block caving method is based on the natural caving of areas of sufficient size when the bottom portion is removed by undercutting. The ore caves and crushes, by its own weight and the weight of the overlying capping, into pieces of suitable size for handling. Drawing of the caved ore at the bottom of the ore column causes the cave action to propagate upwards. When properly applied, block caving results in a lower mining cost per ton than any other underground method.

The method has been applied to the large, massive porphyry copper and iron ore (e.g. Cleveland-Cliff Iron Co. in the Lake Superior district and Bethlehem, U.S.A.) deposits. It has been used in large bodies of chrysotile asbestos (e.g. King mine in Rhodesia) and low-grade disseminated nickel deposits (e.g. Creighton mine in Canada) in Archean ultramafics. Some Kimberlite pipes (e.g. Bultfontein mine in South Africa) and some stratiform deposits in the Copperbelt (e.g. Chingola) have been mined by block caving as well.

In recent years the introduction of LHD loading units has changed the design of the production level, increasing the productivity and lowering the development costs. The use of LHDs eliminates the necessity of complex systems of grizzly levels and ore-passes between the drawpoints and the haulage level.

The application of block caving

The application of block caving requires ores with a suitable fracture pattern to cave easily, and soft enough to break by weight load after caving and by attrition during the downwards flow of the broken ore; ores very soft are not suitable since they will compact again, after caving, by the weight of the overlying material. Ideally the ore must break to small sizes (more than 50% under 1.5 m), whereas the waste must break to coarser sizes to minimize dilution (waste

breaking into fine fragments will flow downwards faster, sifting into
the ore or channelling through the ore, thus increasing the dilution).
The cavability characteristics of the ore bodies must allow the safe
development of undercut and production levels, and at the same time make
the ore cave readily during extraction.   The capping must cave as well
when the underlying ore is dropped, because its weight is essential to
aid in crushing the ore.

The distribution of the values in the ore must be fairly uniform.
Block caving is not a selective method; sorting underground is difficult
and pockets of waste within the orebody are going to dilute the above-
cutoff grades.   On the other hand, an orebody consisting of very rich
pockets in a low grade background could be high-graded, by mining only
the rich pockets using more selective methods, and perhaps improving
the economics of the project.

The orebodies must have enough horizontal area to cave freely
and without excessive dilution from the sidewalls, and a minimum height
for the method to be more profitable than other alternative methods
after carrying the costs of development.   The costs of development are
inversely proportional to the block height.   Irregularities in the shape
of the orebodies on a horizontal plan, increase considerably the dilution
when they are of sizes smaller than the minimum horizontal area to
produce caving (30-40 m x 30 - 40 m); selectivity is only possible at a
larger scale.   The outlines of the orebody should be fairly regular:
small extensions of ore into the walls are not recovered, and pockets
of waste in the orebody cannot be left unmined.

Large massive deposits meet these conditions.   The stockwork-like
pattern of veinlets in porphyry copper deposits and in chrysotile asbestos
in ultramafics makes them easy to cave.   Vein-like deposits must be
wide enough to cave (over 30 to 40 m wide ?) and dip over 65° to have
gravity flow.   Block caving may be used in flat beds if thick enough to
warrant the cost of development; usually the minimum thickness is 20 to
25 m.

The advantages of block caving are:

(i)   its low mining cost (little drilling, blasting and
        timbering are done per ton of ore, and the amount of
        development work per ton is small)

(ii)   a high rate of production is obtained once caving started;

(iii)   centralized production permits efficient supervision
        resulting in higher man productivity and a good control
        of safe working conditions;

(iv)   conditions can be standarized, making for safe and
        efficient operation;

(v)   control of ventilation is less complex compared to other
        mining systems;   and

(vi)   the system is suitable to orebodies of relatively low
        grade.

The disadvantages of block caving are:

(i)   the large capital expense and long development time;

(ii)   maintaining drifts in the draw area can be costly and
        will interfere with production;

(iii)   varying production under increased demand for the product
        requires a long time interval to prepare additional blocks
        for production;

(iv)   to stop the ore drawing for a considerable time may result
        in a complete loss of the development openings;

(v)   ore recovery can be low due to adverse conditions;

(vi)   there is no chance of selective mining of high and low
        grade ore - it must be taken as it comes;

(vii)   the method is inflexible once started - a change to another
        underground method is difficult;   and

(viii) because subsidence normally reaches the surface rain and
        other surface waters can be a serious problem to underground
        workings.

When compared with sublevel caving, block caving has a lower mining
cost per ton, gives a larger daily output from a given area, requires
less development work per ton, and makes easier the control of ventilation.

Sublevel caving is possible in hard ores and smaller orebodies, but it should be considered only where gravity caving might be question-able because of sparse fracturing, or where large blocks might result in high secondary blasting costs.

Cavability

The cavability of the ore is an essential condition to the application of block caving. The rock mechanics properties of the ore must be such that it can be supported while the blocks are being developed and undercut, but it breaks up readily when caved. The orebody should break into sizes that can be handled at the draw-points.

The diagram in Fig. 6.1, taken from Coates (1967; in: Roberts, 1981), illustrates the variations of roof stresses with the height of caving or arching. It can be seen that for a height (c) above the undercut roughly equal to 1.1 times the span (B) the stress σ_t equals zero; arching takes place in the roof rocks and all the stresses are concentrated on the sides of the opening (σ_c). The fill of the undercut and caved void by broken material somehow modifies the distribution of stresses, but over-drawing of that material increases the risk of arching. Hanging-wall rocks within this arch, i.e. with a lower c/B ratio, are under tension (negative values for σ_t).

Because of the tensional conditions above the undercut, strong rock may subside as an integral block. Fracture and fragmentation occur in the lower portions of this block under compressive stress; as the block settles after being undercut, the concentration of compressive stress at random points of contact within the subsiding mass causes the rock to break up by shearing and crushing. In weak ground (see Fig. 6.2) the tension produces fracturing and caving of the roof rocks within the arch.
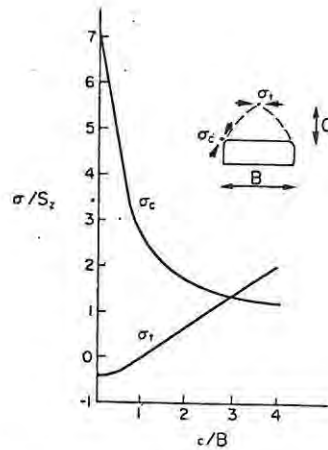
(Roberts,1981)

FIG.6.1
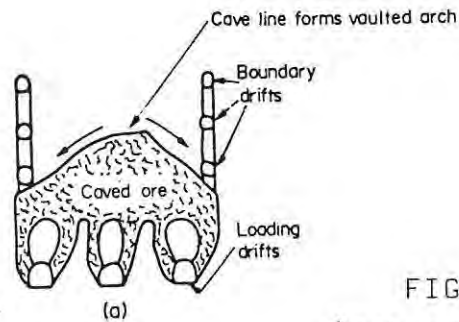
FIG.6.2

(Roberts,1981)
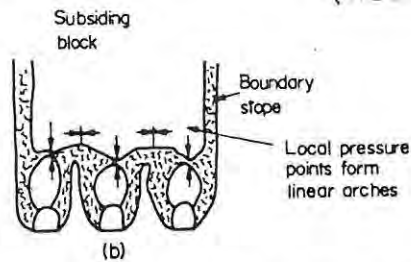
Fracturing produced by block caving: (a) in weak ground; (b) in strong rock
(Woodruff [11]).

The horizontal stress field that exists in a rock mass, can influence cavability by locking rock blocks together and cancelling the tensional forces above the undercut area (that would otherwise favour caving).

The broken material must be drawn off as fast as it is produced; if not it will be compacted again by the weight of the overlying block of strong rock.

In the general case the pressure arch that is imagined to
exist above an excavation will in reality be a three-dimensional
dome.   Relationships between span and height of the dome have
been studied by Denkhaus (1964; in: Roberts, 1981).

The diagram in Fig. 6.1. also shows that the increase in
span increases the tensional stresses on roof rocks.   If these
rocks are sufficiently cohesive, a minimum span must be reached to
produce caving, but with further increases in the span the material
within the core may support itself by forming smaller domes within
the main dome.   At Climax (Gould, 1982), successful caving action
requires an undercut area of approximately 122 m by 122 m.

The intensity of the fracture pattern is a critical parameter
analyzed to determine the cavability.   Several sets of fractures
are essential to promote good caving;   ideally, two vertical sets at
nearly right angles to each other and a third set nearly horizontal
are required to insure a good caving orebody.   The more closely
spaced the fractures the more readily the ore will cave.

Nearly horizontal fractures would probably have the same response
of a horizontally bedded deposit, and even though the ore will break
in tension or shear, the blocks (slabs) would probably be too large
to handle through the draw-points and would certainly result in high
secondary blasting costs.   To avoid high secondary blasting costs,
at least 50% of the ore should break to 1,5 m dimensions or less.
The highest secondary blasting costs seem to be incurred during the
first 30% of the draw, when breaking depends almost entirely on the
action of gravity on the fracture planes;   in later stages, attrition
helps to reduce the size of the ore fragments.

The fracture patterns in an orebody can be evaluated from
diamond drill cores and studying the development openings in the
orebody.   When recording the structural information from drill cores,
the spacing of fractures, their attitude to the drill hole, and the
type of fracture filling (indicator of the shearing strength along the
fracture) must be logged as carefully and consistently as possible.

The parameter most widely used to determine cavability of a rock mass is the rock-quality designation (RQD). It is based on the percentage of core recovered counting only the pieces of intact core 10 cm or longer. An alternative method consists in substracting of the total drilled the percentage of core lost and the percentage of those fragments, less than 10 cm long, that broke along a geologic feature. In any case, all fresh conchoidal fractures, separations due to disking in the hole, and any other fractures due to operation of the drill machine should be ignored. Rock-quality can be divided in different classes:

RQD, %	Quality
0- 25	very poor
25- 50	poor
50- 75	fair
75- 90	excellent

and is estimated as shown in the following example:

total length drilled	=	130 cm
total core recovered	=	104 cm
core recovery	=	104/130 = 80%
added core lengths greater than 10 cm	=	71.5 cm
RQD	=	71.5/130 = 55%

Fig. 6.3. shows the distribution of different classes of ground conditions, ranging from 1 (very good) to 5 (very poor), on a level plan of the King chrysotile asbestos mine in Rhodesia (Ferguson, 1979). These plans are used in the assessment of mining methods, support requirements, siting major excavations, and definition of cave angles.

(Ferguson, 1979)   FIG.6.3
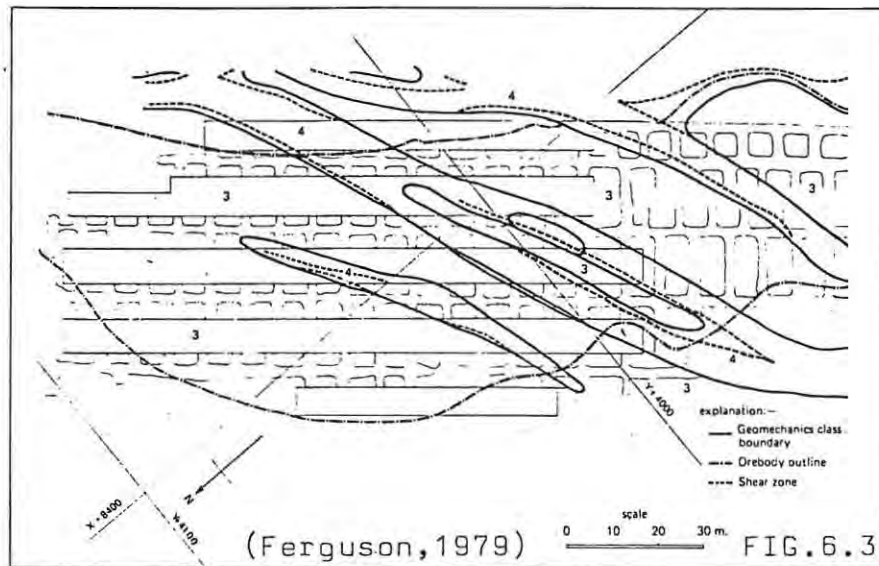
King Mine, Block W-2, a geological and geomechanics plan of 192 level.

(When using RQD, it must be remembered that drill holes are directional and can entirely miss important fracture trends. It is important to drill in different directions).

Layout

Three distinct forms of block caving have been used in the past. The first consists in dividing the horizontal area into rectangular or preferably square or nearly square blocks, and drawing evenly over the entire area to maintain an approximately horizontal plane of contact between broken ore and caved capping. Panel caving consists in dividing the horizontal area into panels across the orebody, retreating by undercutting manegeable areas from one end of the panel to the other, and maintaining an inclined plane of contact between broken ore and caved capping; block panel caving, a combination of these two forms of caving is frequently in use. The third form, the mass caving, has no division of the horizontal area of the orebody into definite blocks or panels, and is normally used in pipe-like bodies.

The main factors affecting the layout design are the height of the blocks, the horizontal area, the rate of draw, the drawpoint spacing and the weight problems.

High-column blocks are desirable, since the development cost
per ton of ore is inversely proportional to the column height.
The maximum practical height of the blocks above the extraction level
depends on the total height of the deposit, the dip of the orebody,
and the character of the ore and capping.   Capping breaking to much
finer fragment sizes than the ore increases the dilution in high
blocks, because the fine waste tends to flow faster and intermix
with the ore; when the capping breaks into coarse fragments, the
dilution problem is greatly reduced.   The table in Fig. 6.4. shows
the height of blocks, as well as horizontal block sizes, in different
mines using block caving.

Block size in Block Caving*

Mine	Type of Deposit	Block Size, m	Height, m
Thetford Mines, Asbestos Corp. Ltd.	Asbestos igneous stockwork	49 x 49	122 and less
Miami Copper Co., Tennessee Corp.	Porphyry copper	30 x 49	183
El Salvador, CODELCO-Chile	Porphyry copper	29 x 36 to 48 x 74	49–213
Mather, Cleveland-Cliffs Iron Co.	Hematite	27 x 27	61–76
Creighton, International Nickel Co.	Disseminated norite	Mass cave	152
San Manuel, Magma Copper Co.	Disseminated chalcocite Altered granite	Panel cave	183 and greater
Grace, Bethlehem Steel Corp.	Massive magnetite	46 x 46	34
Cornwall Nos. 3 and 4, Bethlehem Steel Corp.	Magnetite	Panel cave	30–61
Climax, AMAX	Disseminated molybdenite	Mass cave	91
Urad, AMAX	Disseminated molybdenite	Mass cave	91
Dé Beers Cons. Mines, Ltd.	Porphyritic volcanic stock	Panel cave	122–183
El Teniente, CODELCO-Chile	Massive sulfide copper	60 x 90	100–180
Henderson, AMAX	Disseminated molybdenite	Mass cave	122

*Source: SME Mining Engineering Handbook, 1973.

FIG.6.4 (Tobie & Julin,1982)

The horizontal block area should be large enough to cause the
ground to cave with relative ease, but small enough not to exert
excessive weight on the extraction openings.   The best horizontal
section for the blocks is a square one; a rectangular area still may
not cave even though a very long cave is opened up because arching
will occur across the shorter dimension.   The horizontal block area
should be limited to that in which the broken ore can be drawn at the
same rate that the ore is broken by caving.

The caved material should be drawn fast enough to avoid packing
of the broken ore and permit the caving to proceed upwards through the
rock mass.   Depending on the cavability of the ore, the rate of drawing
varies from the tonnage corresponding to the caving of a slice 152 mm
high to that of an 1.2 m high one per 24-hr day.   If the ore is drawn
faster than the caving progresses, dangerous voids will be produced;
they could result in dangerous air blasts on collapse.

The spacing between drawpoints is a function of the size of the
fragments to be drawn.   The more finely the rock breaks, the closer
the draw point spacing.   The zone of disturbance of one draw point
should slightly overlap all adjacent zones of disturbance so as not
to leave pillars of ore that do not move when the ore is drawn;   the
flow of broken ore is discussed in the section on sublevel caving.
An interesting paper on the design of bins for reliable flow by
Jenike and Johanson (1972) contains the graph on Fig. 6.6, relating
the hopper slope angle to the angle of internal friction (a function
of the particle size and shape, and the moisture - among other factors
that can be measured in a Flow-factor Tester (op. cit)) in order to
obtain the desired mass-flow.   The principles (see Figs. 6.6, 6.7 and
6.8) could be applied to draw-point design.   The spacing between draw
points should not exceed 15 m x 15 m, and is best determined by experience
in other mines working under similar conditions and by experimentation;
the table in Fig. 6.5 shows some typical spacings as a function of the
fragment size.

Drawpoint Spacing*

Mine	% of Ore Size, ft†				Drawpoint.Spacing, m
	+5	2-3	1-2	-1	
Creighton	30	30	40	—	9 x 12
Thetford	20	25	25	30	7.6 x 7.6
Mather	5	10	15	70	4 x 8.5
Miami Copper	—	—	—	—	5 x 5.7
El Salvador		Varies			43.6—148.6‡
San Manuel		Fine ore			4.6 x 5.3
Climax	7	24	23	46	10 x 10.4
Grace	50	20	20	10	6 x 9
Cornwall	10	20	20	50	7.6 x 12
Urad	40	15	15	30	9 x 9
Henderson	—	—	—	—	12 x 12
De Beers	—	—	—	—	4.6 x 6.8

* Source: *SME Mining Engineering Handbook*, 1973.
† Metric equivalent: ft x 0.3048 = m.
‡ 43.6 m for fine ore; 148.6 m for coarse ore.

FIG.6.5
(Tobie & Julin,1982)

As ore is removed and voids are created, the weight of overlying
rock that has not caved is transmitted to the perimeter of uncaved ore
(σ_c stresses in Fig. 6.1), causing excessive damage on fringe workings.
The configuration of the advancing cave face can help to minimize the
effects of this transferred weight (see Figs. 6.17, 6.25 and 6.26).
Within the active caved area, weight should be avoided by opening up
only sufficient area to maintain the desired production rate, drawing
steadily and evenly, and removing any remnants during undercutting
that could act as pillars.
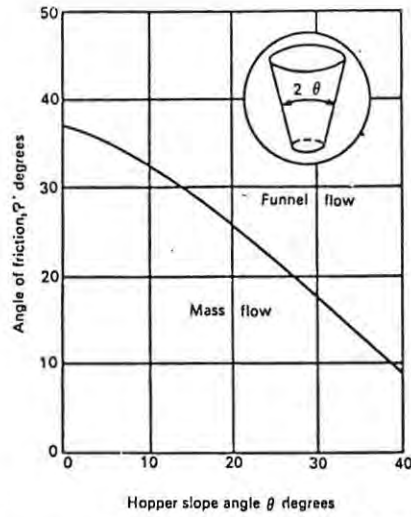
FIG.6.6

Ranges of mass flow and funnel flow in conical hoppers

First-in,
last-out,
segregates,
ratholes,
flushes.

FIG.6.7

Funnel-flow bin

First-in,
first-out,
deaerates,
remixes,
feeds uniformly
at constant
density.

FIG.6.8
(Jenike &
Johanson,1972)

Mass-flow bin
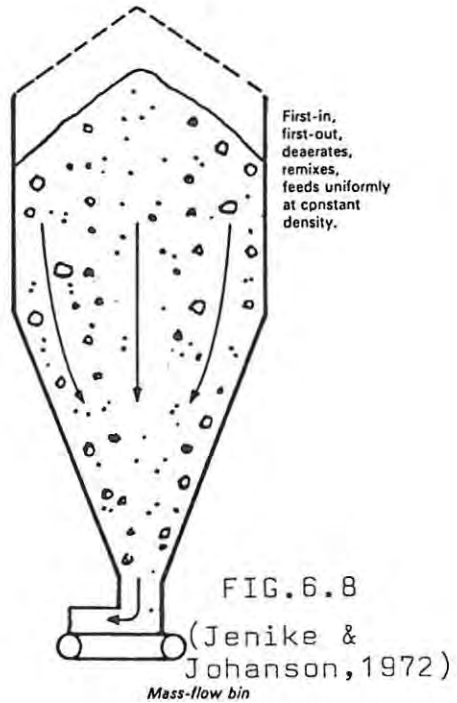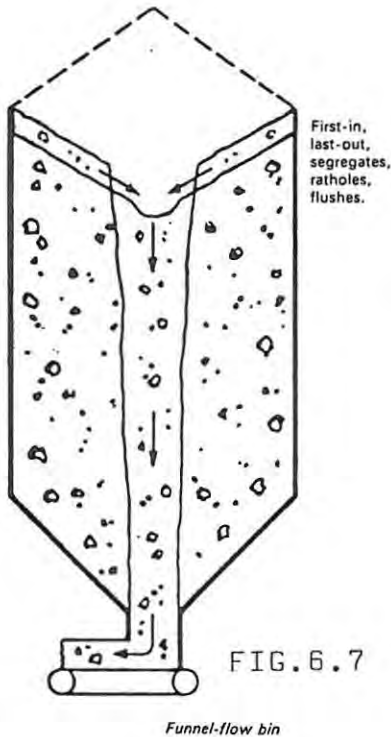
Fig. 6.9 shows a typical block caving layout for an orebody divided
in square or nearly square blocks. A similar layout for a deposit with
more than one production level is shown in Fig. 6.10. In both examples

the ore from the drawpoints moves by gravity through a system of
ore-passes and grizzly levels into loading chutes in a haulage level.
An alternative method using LHD equipment is shown in Fig. 6.11 (El
Teinente porphyry copper mine, Chile) : it eliminates finger raises,
ore-passes, grizzly levels and box-loading.
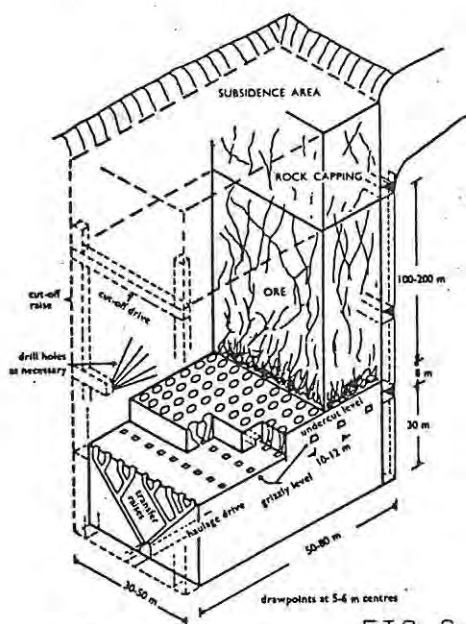
Typical block caving layout

FIG.6.9

(Thomas,1978)
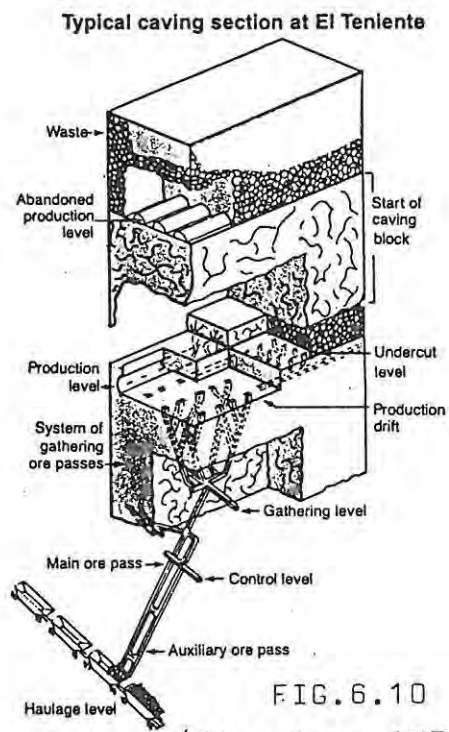
Typical caving section at El Teniente

FIG.6.10

(Sisselman,1978)

At Henderson mine (Doepken, 1982), LHD equipment is in use and
the layout of the cross-cuts to the drawpoints in the production level
has been modified to a new "Z"-style pattern, so as to have both entries
parallel to the principal horizontal stress (see Fig. 6.12).   From the
production levels the ore is transferred through a system of ore-passes
to the main haulage level (see Fig. 6.13).

Alternative mining method using LHD equipment

Section A-A

FIG.6.11
(Sisselman,1978)
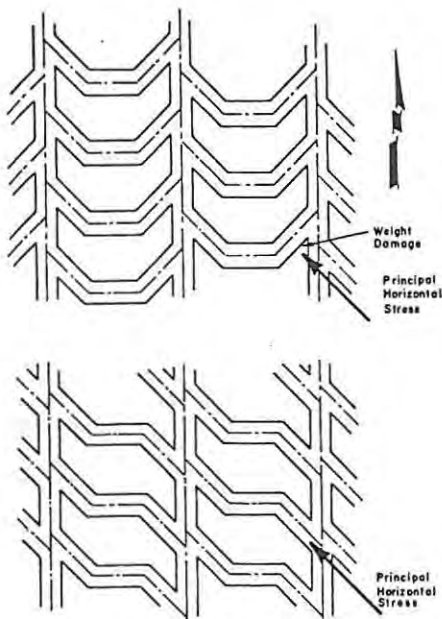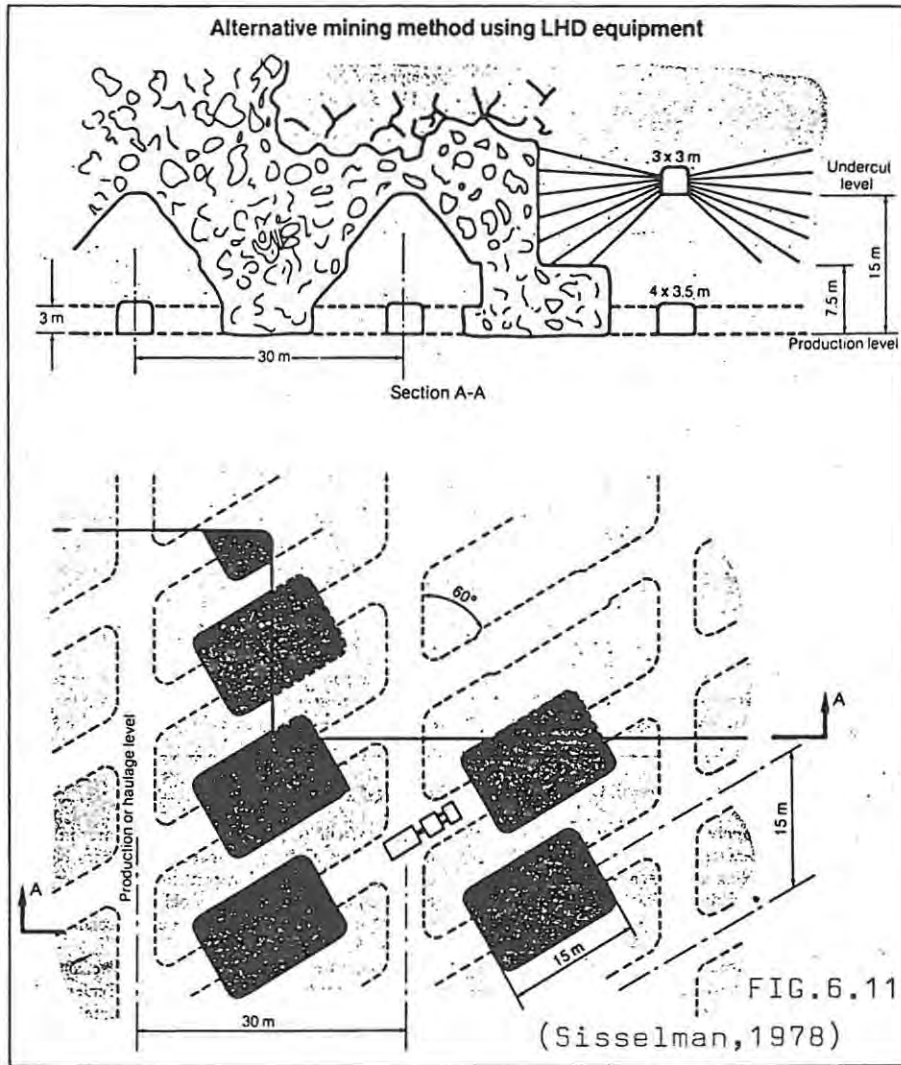
Drawpoint configuration. Top is old type drawpoint,
bottom is new type drawpoint.

FIG.6.12   (Doepken,1982)

Isometric of mining levels, Henderson mine.
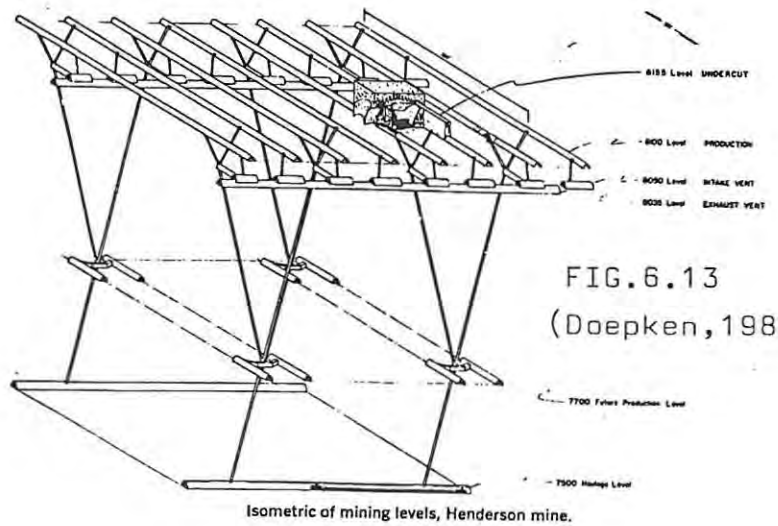
FIG.6.13
(Doepken,1982)

The advantages of LHD mucking over the traditional "manual" mucking are that it eliminates the need for ore-pass connections to each draw point, has a greater productivity and allows for a larger production rate; the drawpoints can be larger making easier the secondary blasting, and the LHD buckets can handle larger chunks and have an undercutting action on the broken ore at the draw point. The main disadvantage is the increase in size of the adits and draw point spacing to accomodate the LHD units, resulting in a poor confinement of the crown-pillar rock mass. The design of the production level, or draw point horizon, must have a reduced number and size of openings, which is achieved - for example - with staggered (and not opposite to each other) draw point - crosscuts, and in an angle to accommodate the turning circle of an LHD. A reduced crown-pillar height makes easier to break up the large rocks producing hang-ups.

FIG.6.14   Present method of block caving (Sisselman, 1978)
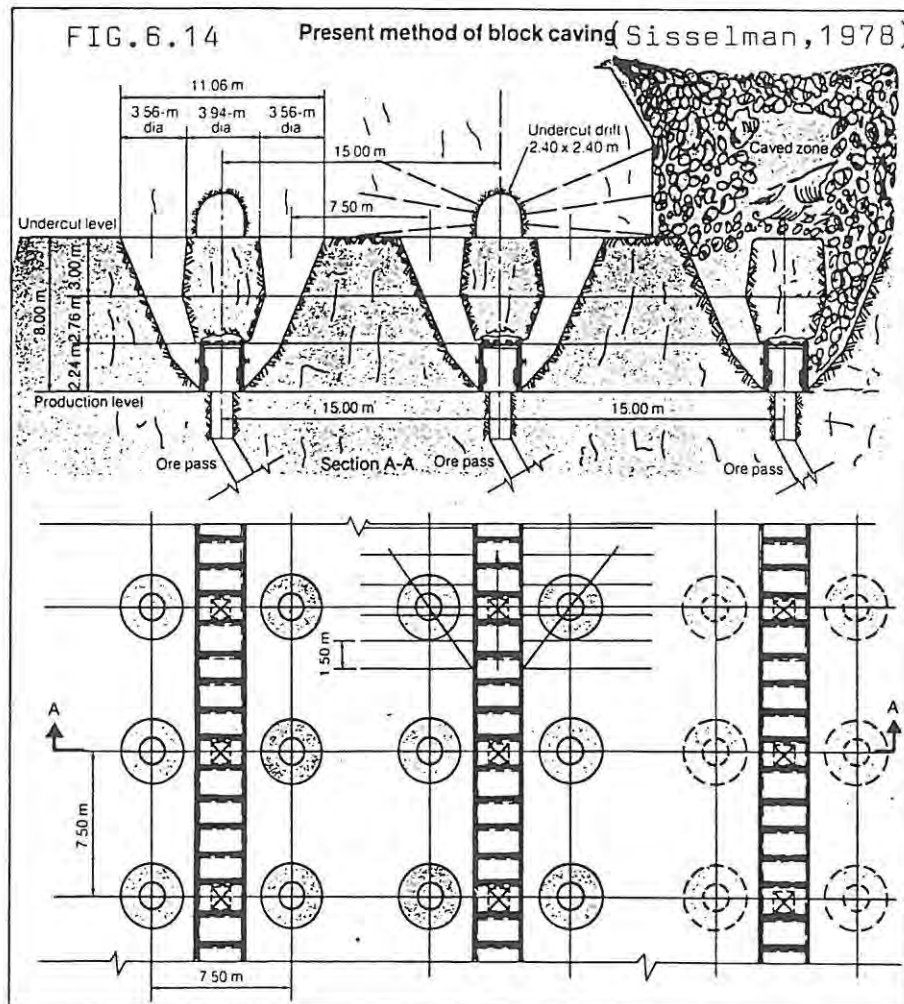
Undercutting in a "gravity"-system is done in the way illustrated in Fig. 6.14, in use at El Teniente, while in a LHD-operation it is done in the way shown in Fig. 6.11 (similar to the "old style" at Henderson) or in the way shown at the bottom of Fig. 6.15 which corresponds to the "now style" at Henderson.
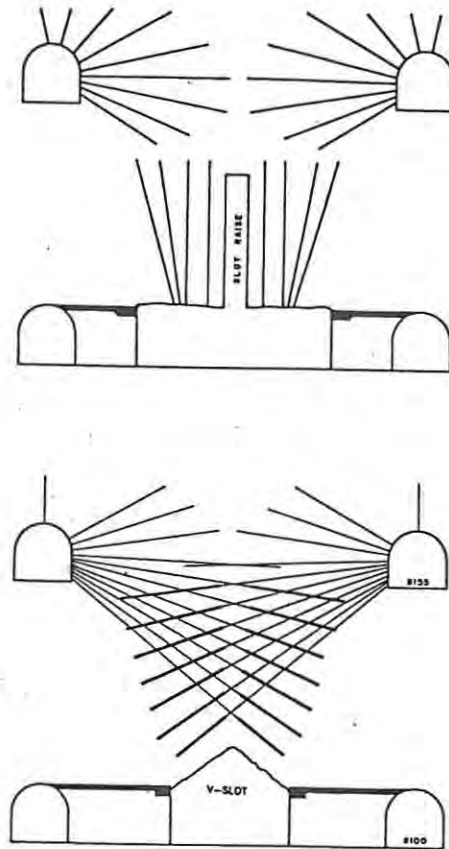
Draw-bell drilling. Top is old style bell development, bottom is new style bell development.
FIG.6.15 (Doepken,1982)

The undercutting in a block always retreats moving away from a previously undercut block, as seen in Fig. 6.16, to avoid dangerous situations same as the one at Shabanie mine (Ferguson, 1979) shown in Figs. 6.17 and 6.18; previously mined blocks transmit stresses to the perimeter of uncaved ore.

The block can be presplit along its sides using blastholes from boundary drifts and corner raises, as seen in Fig. 6.9, or with vertical blastholes from an upper production level or from the topmost boundary drift.   The presplit blastholes are shot simultaneously to maximize the effect (mostly shock wave action and little or no gas effect, because of

Progression of undercutting
(typical block)
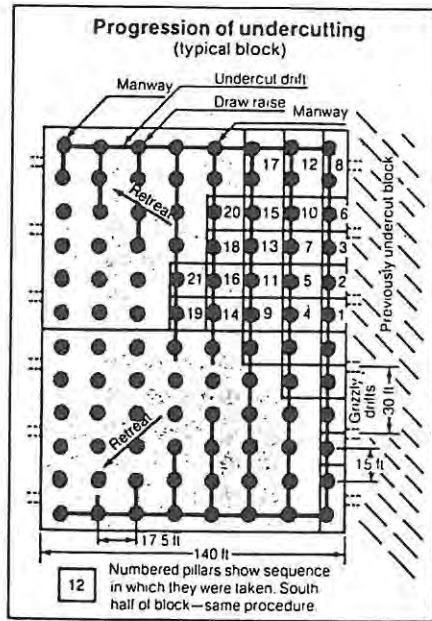
FIG.6.16
(Jackson,
1978)

FIG.6.17
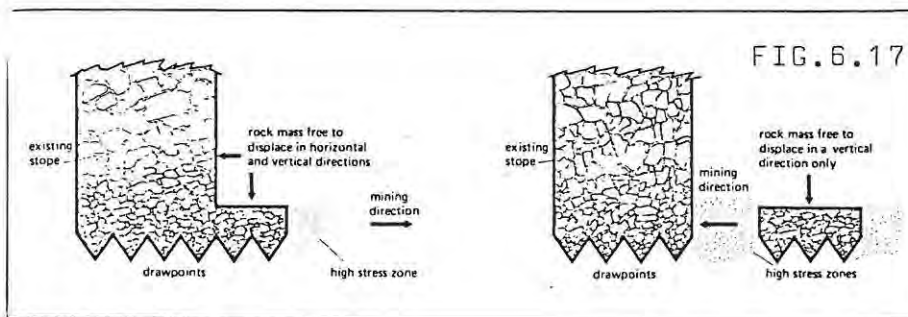
The initiation of caving of a block cave stope adjacent to an existing block cave stope. (left): The preferred mining direction of an undercut excavation adjacent to an existing block cave stope. (right): An unfavourable mining direction of an undercut excavation adjacent to an existing block cave stope.

(Ferguson,1979)

FIG.6.18
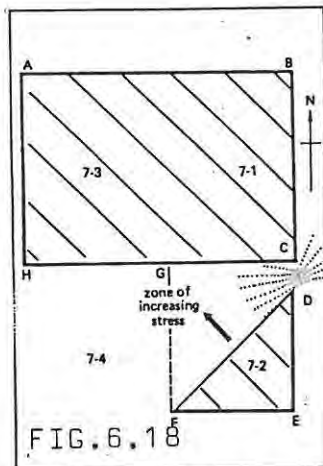
The outline of the cave stopes of blocks 7-1, 7-2 and 7-3 Shabanie Mine.

Mining Magazine — February 1979

the confinement).   To help the initial caving or to stop the
caving at a cutoff boundary, a slot stope or boundary stope
could be necessary along one or more sides of the block (see Fig. 6.1);
a shrinkage stope along a cutoff boundary has the advantage of more
selective mining of the more irregularly distributed mineralization.
Ground with a good RQD would need additional blasting to help the
caving;   these blastholes can be drilled from boundary drives (see
Fig. 6.9) or, as in use at El Teniente in the zone of primary copper
sulphides, vertically downwards from a drilling level (see Fig. 6.19).
A similar practice of induced block caving in Russia is described by
Gaidin (1971), and is illustrated in Fig. 6.20.

Fig. 6.21 shows the general sequence followed in panel caving by
blocks; panel caving (s.s.) is similar, but undercutting proceeds
continuously (see example at Climax in Fig. 6.22) and the horizontal
area of the orebody is not divided in blocks, and no rib-pillars are
left between the blocks.   At Climax the ore is undercut and caved
into the finger raises, scraped towards the drawhole and loaded into
the cars waiting in the haulage drift (see Figs. 6.23 and 6.24).

FIG.6.19
(Sisselman, 1978)

FIG.6.20
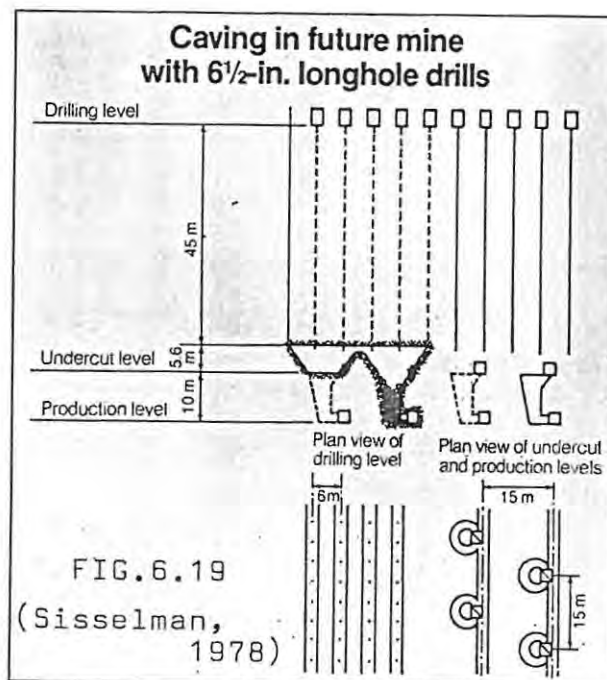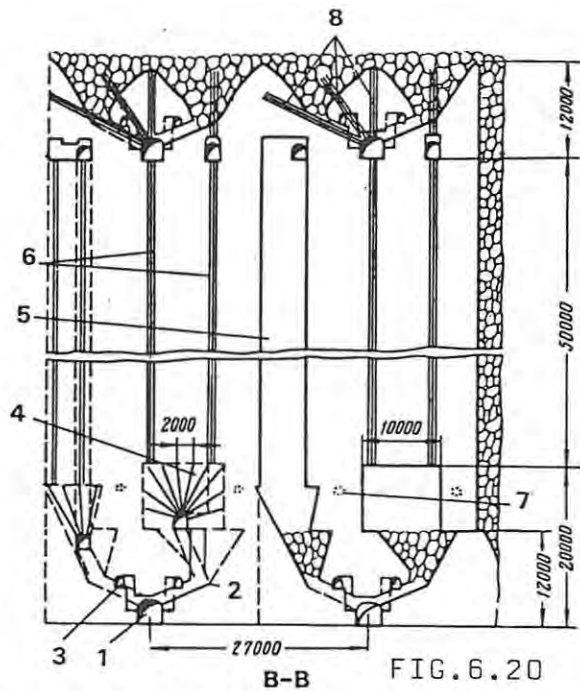
Continuous induced block caving with flow-line vibratory ore drawing.
(1) haulage drift; (2) chambers to mount vibration rigs; (3) inspection
workings; (4) high undercut; (5) compensation slot; (6) bunches of deep
boreholes; (7) horizontal bunches; (8) boreholes

Mining Magazine—August 1971, Vol. 125, No. 2 (Gaidin,1971)
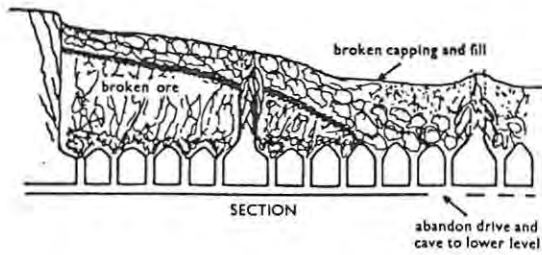
FIG.6.22
(Julin et
al.,1973)
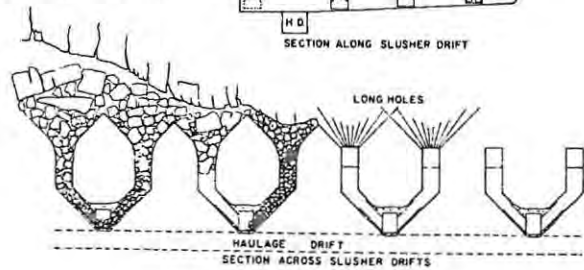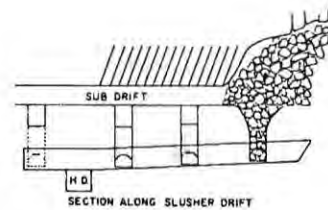
Long-holing and blasting sequence, Climax Molybdenum.
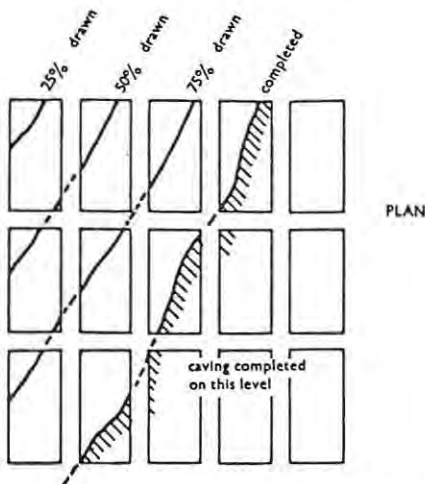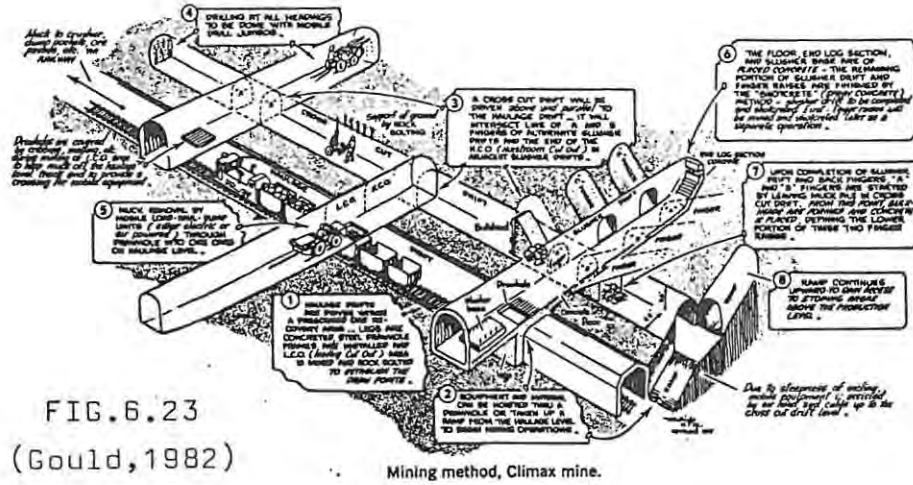

Draw control for block caving

FIG.6.21 (Thomas,1978)

FIG.6.23

(Gould,1982)

Mining method, Climax mine.
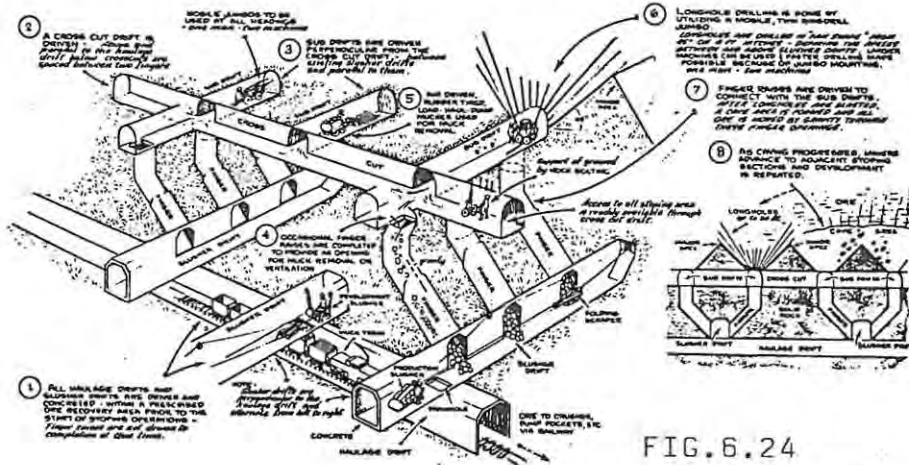
FIG.6.24

(Gould,1982)

Undercutting method, Climax mine.

In panel caving, the layout of the production level can be designed to use LHD units in a way similar to the one shown in Figs. 6.11 and 6.12. To avoid excessive stresses over the periphery of the undercut area being drawn, drawing and undercutting retreat in a direction diagonal to that of the blocks (block panel caving) as it is shown in Fig. 6.21. For the same purpose, the blocks in panel caving at the San Manuel porphyry copper mine (Tobie et al., 1982) were originally undercut in the sequence shown in Fig. 6.25, but the new diagonal retreat shown in Fig. 6.26 has given better results; retreat in each individual block proceeds in a diagonal direction as well, going away from the already undercut area.
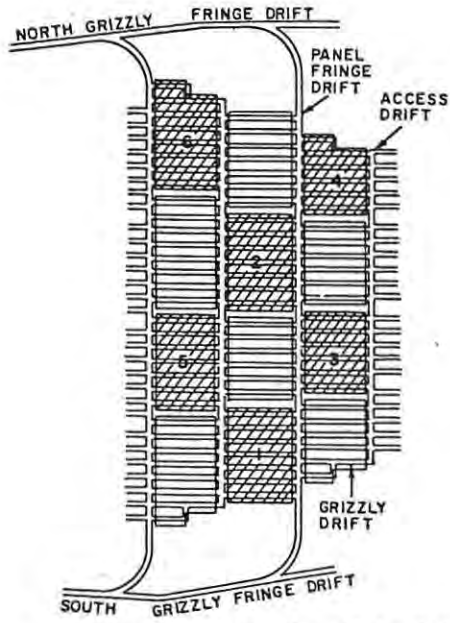
Sequence of block undercutting, original method.

FIG.6.25 (Tobie,Thomas
& Richards,1982)
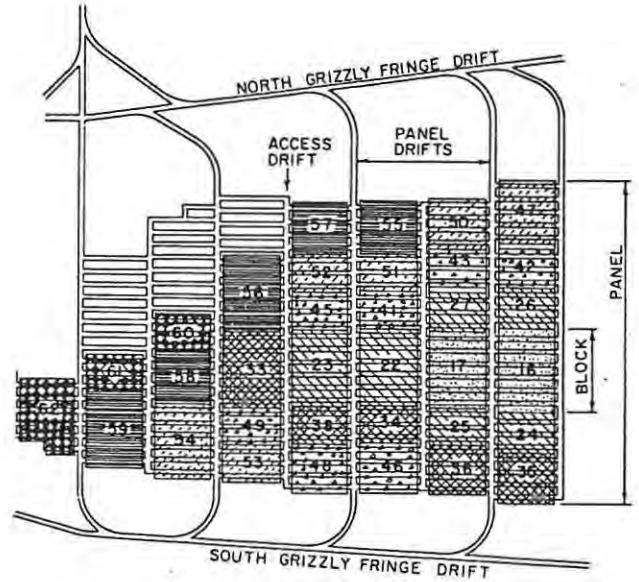
FIG.6.26 (Tobie,Thomas &
Richards,1982)

Diagonal retreat panel caving
by blocks.
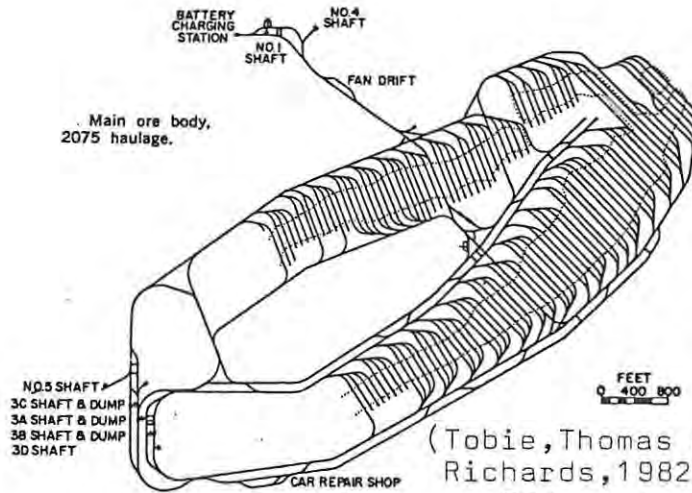
(Tobie,Thomas &
Richards,1982)
FIG.6.27
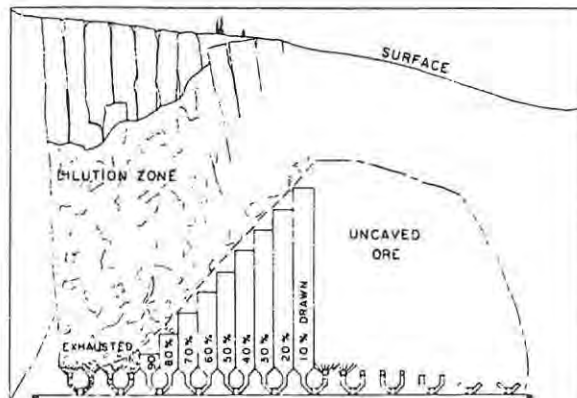
Dilution control, Climax Molybdenum.
FIG.6.28   (Julin et al.,1973)

The cave line is advanced across the haulage drifts in an optimum, stepped, concave front across the entire width of the orebody, thus obtaining the most favourable distribution of stresses. Large irregularities in the cave line cause dramatic increases in weight concentrations and serious drift and draw point damage.

The draw control in panel caving is essential to cut dilution. Drawing evenly and having a nearly horizontal contact between the caved ore and waste minimizes the dilution, but it requires a larger undercut to produce a given output, with the consequent weight problems in the production levels and meaning a high working capital. A nearly vertical slope would minimize the weight on the drifts and the costs of maintenance, but it would increase dilution to an impossible maximum. Between these two limits an angle of 60° is frequently adopted; it produces moderate dilution and permits a caving area large enough to yield the scheduled daily tonnage without excessive repair costs (see Figs. 6.21 and 6.28).   At Humboldt mine (Peele, 1941) the slope between broken ore and waste ("angle of retreat") averaged 60°, but when the weight problems on the workings grew excessive the drawing area was reduced by steepening the slope to 70°.   The draw control has obvious grade-control implications which are discussed in this same chapter.

On an overall scale, Fig. 6.27 (San Manuel porphyry copper mine, USA) is an example of the developments of an haulage level for the orebody shown there.   Since the disturbances due to the caving will extend out to a surface going from the edges of the undercut area to the surface with a slope of roughly 45°, the shafts have been placed accordingly in undisturbed zones.   Shafts No. 1 and No. 4 are used for transport of men and materials, and to provide intake ventilation, and were initially used to hoist the muck from the developments; their central position reduced the costs of tramming that muchk.   During the production stage, the shafts 3A, 3B, 3C and 3D provide muck-hoisting capacity, and exhaust ventilation capacity sufficient to balance the intake; their location optimizes the transport from both branches (transport towards the shafts No. 1 and No. 4 from the south branch has to go through the obstacle of the workings on

the northern branch of the orebody). On the northeastern side the orebodies come together and the area undercut is much larger; the disturbances of caving extend further away from the orebodies and if the hoisting shafts were located on that side, the pillar should have been wider and the tramming distances much longer. The general direction of retreat is towards the shafts, i.e. towards the southwest. A detail of the developments and mining sequence is seen in Fig. 6.26.

Evaluation and grade control

Orebodies with an irregular shape in horizontal section, cannot be mined selectively (using block caving) at a scale smaller than that of the horizontal block size, which, in turn, is limited by a minimum undercut area necessary to produce the caving. Changes in shape of the orebodies at a scale larger than that minimum undercut area required, can be followed by the mining limits, as it is shown in Figs. 6.25, 6.26 and 6.27.

In a vertical direction, the caving of blocks maintaining an even horizontal caving surface requires a fairly flat upper limit of the ore. Irregular upper limits of the ore will make necessary to adopt a multi-production level system, to cut the dilution that would result when mining adjacent blocks of highly contrasting block height. In this situation, the distance between production levels will define the scale of selectivity in a vertical direction. The decrease of the distance between production levels increases the selectivity, but it is limited by the consequent increase in costs of development per ton of ore mined.

Pockets of waste within the orebodies can be sorted and hoisted as waste by monitoring the grade at the drawpoints. This waste must be developed, drawn, sorted and hoisted to surface, and these costs make uneconomic the mining of orebodies that require a high rate of sorting; block caving is not a selective method and a fairly uniform distribution of the values in the orebody is necessary.

Vein-like deposits, wide enough to attain caving and dipping
over 65° to have gravity flow, can be developed and mined by "continuous
block caving", in a way similar to that in use at the Grängesberg
iron ore mine in Sweden, illustrated in Fig. 6.29 (note the design for
the use of LHD units).   Ideal conditions are strong sidewalls from
which the ore parts easily; isolating stopes, or presplitting blast-
ing, may be used to free the ore from the walls if their cost is
justified by the ore tonnage thus released.

Block caving may be used in flat bedded deposits if they are
thick-enough to carry the cost of development.   Usually, a minimum
thickness is 20 to 25 m.   An example of panel block caving at Chingola
(Zambian Copperbelt) is shown in Figs. 6.32 and 6.33.
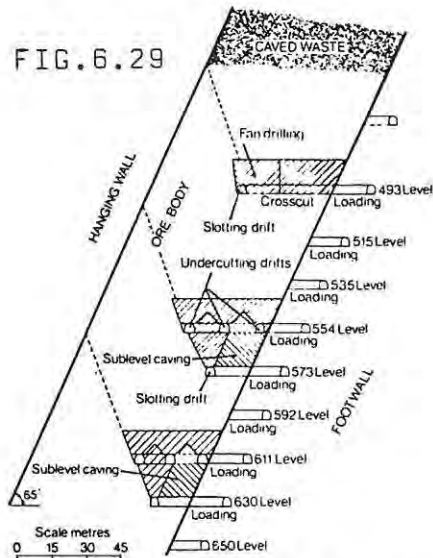
FIG.6.29

Simplified diagram of Grangesberg continuous block caving system
(Min.Ann.Rev.,1976)
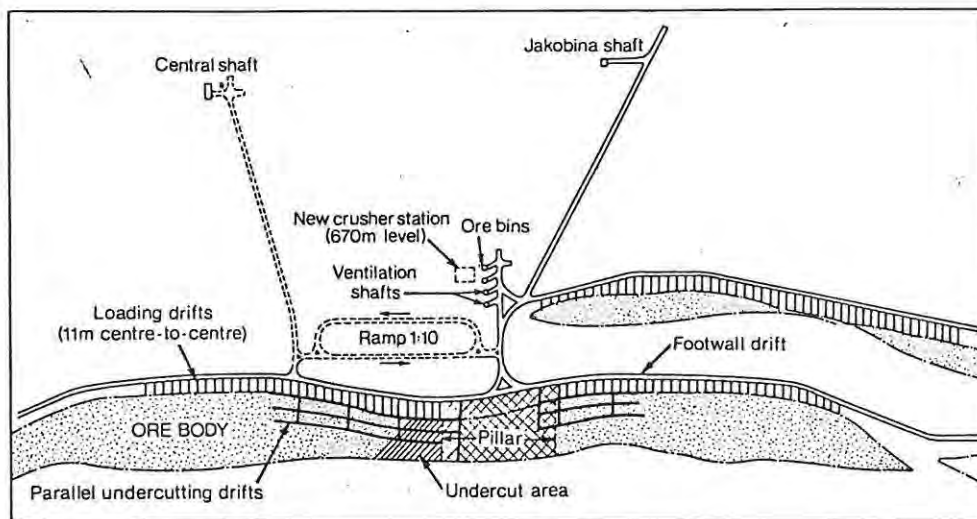
Plan of new continuous block caving system (as at 554 undercut level)
Mining Magazine—June 1975   FIG.6.30

At El Salvador porphyry copper deposit in Chile (Journel and Huijbregts, 1978), for production planning the ore reserves on each block of the multi-production level block caving system are estimated by kriging. Assay data comes from 3 m samples of drill cores in all directions, and from 3 m long continuous channel samples along the drifts. A single geostatistical structure model, isotropic with a nugget and a hole effects, is used for both sets of data. The ore from massive breccia bodies follows the exponential model $(1-e^{-ar})$ and the intense enrichment within fractures has a pseudo-periodic character corresponding to the hole effect $(1- (\sin br)/br)$. The model can be stated as follows:

$$\gamma(h_u, h_v, h_r) = C_0 + C (p(1-e^{-ar}) + (1-p) (1- \frac{\sin br}{BR}))$$
$$\forall r = |h| > \lambda$$

In the kriging of 100 m x 100 m x 100 m mining blocks, the following data sets were used:

(i) the 10 clusters of drill intersections within the horizontal slices of the block, each having a thickness of 10 m.

(ii) the 10 clusters within the 10 slices of the aureole of a 300 m x 300 m x 300 m volume surrounding the block.

(iii) the set of channel samples taken along the drift which run along the base of block B.

This provides a total of 21 weights (λ_i) for the 21 data sets and a stationary kriging system of dimension 22. An example is provided in Fig. 6.31. Note the small weight $\lambda_{21} = 0.055 = 5.5\%$ given to the extremely well sampled level at the base of the block; the number of channel samples (263) was shown to be excessive by studying their influence on the kriging variance σ_k^2. The grades of the 10 horizontal slices of each block were used to plan the draw charts and the development schedule to achieve a fixed mill feed grade through blending; the rate of draw in each block is fixed (determined by the caving rate), and in order to obtain an optimum blending the schedule of developments, i.e. the time of starting the draw of ore from each block, is the main variable.
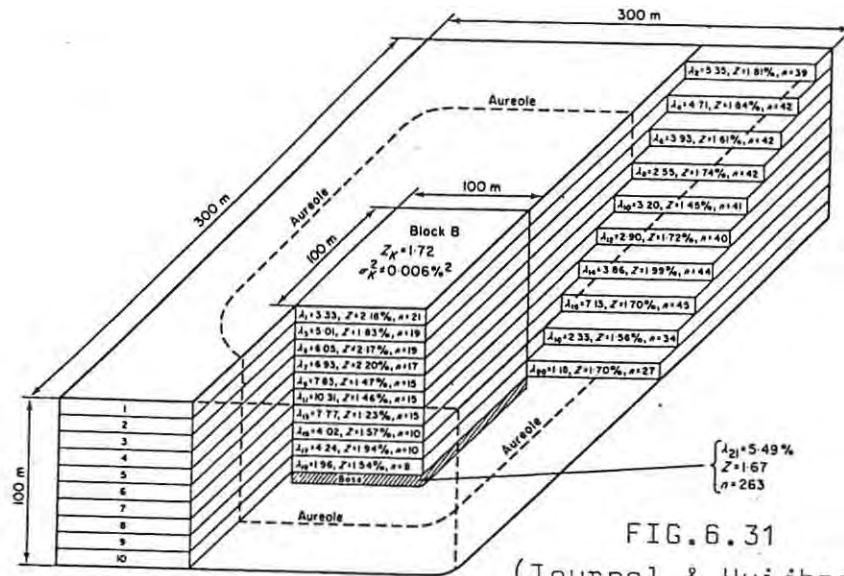
El Salvador's kriging plan. n = number of core samples within each slice, Z = mean grade of each slice, λ = weight in %.

FIG.6.31
(Journel & Huijbregts, 1978)

At the Chingola mine in the Copperbelt (Maxwell, 1979), panel block caving is being applied to shallow-dipping stratiform copper deposits. The caving principles, undercutting methods and haulage system are shown in the diagrams in Figs. 6.32, 6.33 and 6.34. Recoverable ore reserves were originally estimated using geological contour plans of each scraper drift, i.e. in vertical sections parallel to the strike of the orebody each drawn through the axis of a scraper drift (see Figs. 6.32 and 6.33). Accounting for the dilution from overlying waste, tonnage and grade factors were applied to correct the estimates. This method does not reflect the fact that the dilution is minimum during the early productive life of a scraper drift and increases considerably when the draw approaches 100% of the in situ ore tonnage. Consequently, the overall grade and tonnage factors used had extremely large monthly variations (see Fig. 6.35), and they could not be used to predict the grade of the tonnage to be drawn from a scraper drift at a specific stage.

FIG.6.32
(Maxwell,1979)
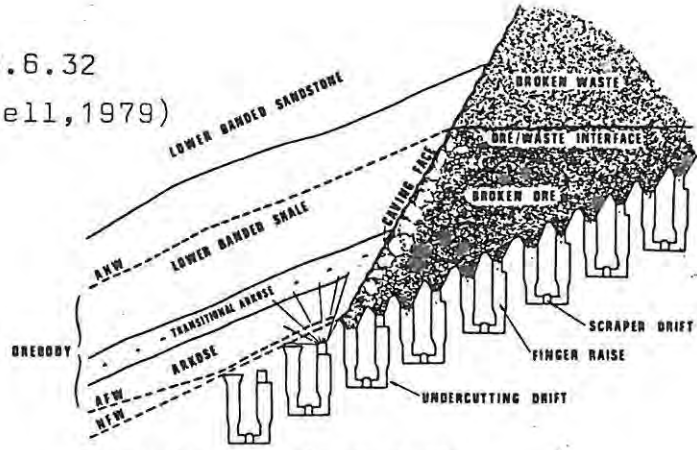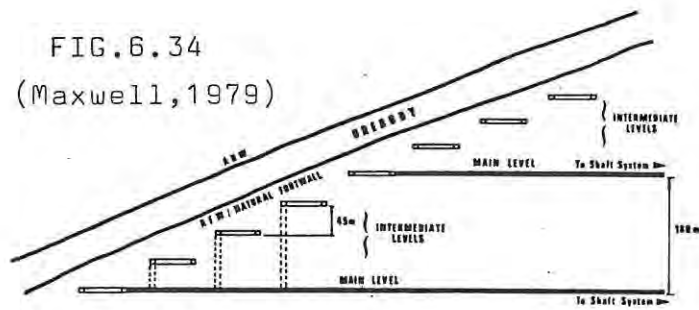
Diagrammatic section illustrating caving principles
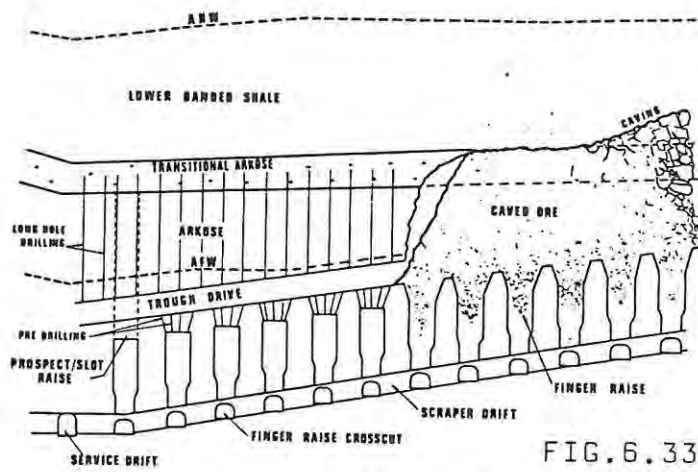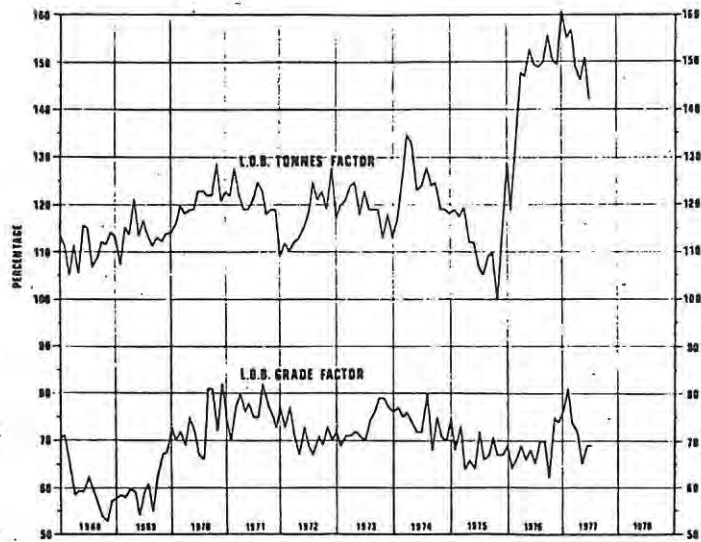
Undercutting (section along scraper drift)
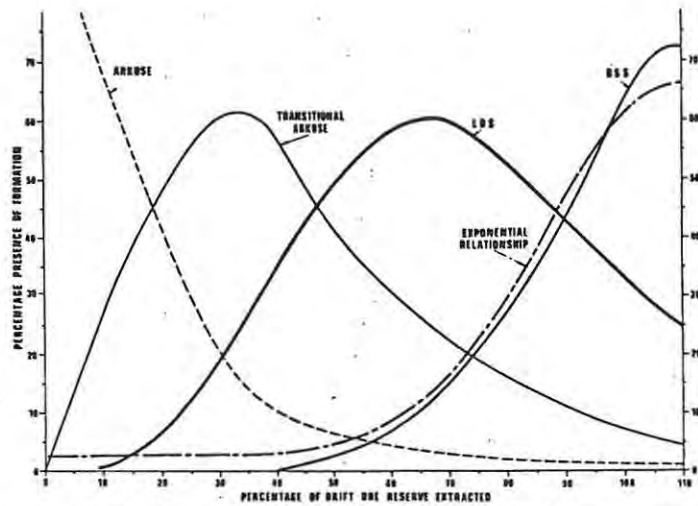
FIG.6.33
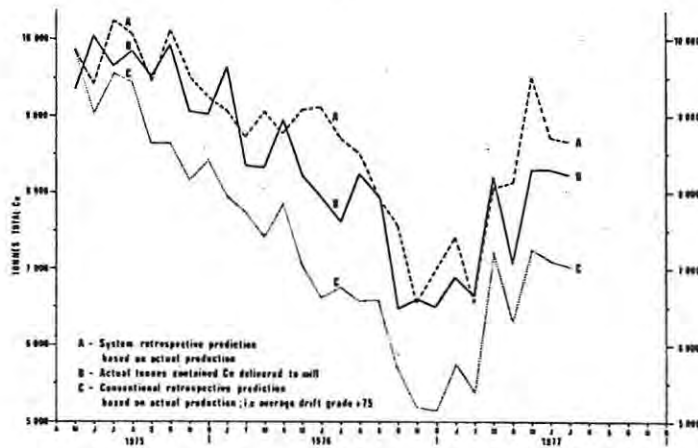(Maxwell,1979)

FIG.6.34
(Maxwell,1979)

Haulage system

Monthly tonnage and grade factors for Lower orebody

(Maxwell,1979)

FIG.6.35

In 1972 a computerized ore reserve system was developed at Chingola. The orebody was divided in vertical slices parallel to the strike, each one representing the ore to be drawn from a specific scraper drift. Grades and tonnages were calculated weighting the borehole intersections by the inverse of the distance of the borehole to the center of the slice; in each slice, the computer could also estimate grades and tonnages for each pair of finger raises. The variation of the tonnage and grade factors, with the increase in the percentage of in situ ore extracted in a scraper drift, was derived from a generalized empirical relationship between the percentage of ore reserve extracted and the percentage-presence of various rock types in the draw (see Fig. 6.36). In Fig. 6.36, the inverse of the "exponential relationship" gives the variable grade factor.

The capacity of these extraction-percentage related tonnage and grade factors to predict the grade of the drawn tonnage at any stage, was tested retrospectively using the records of past ore reserve estimates and mine production. The comparison in Fig. 6.37 (curves A and B; curve C corresponds to the old system of fixed factors) shows good results.

Relationship between percentage of ore reserve extracted and percentage presence
of various rock types

FIG.6.36   (Maxwell,1979)

Comparison of actual tonnes Cu delivered to mill with conventional and system
retrospective predictions

FIG.6.37   (Maxwell,1979)

The conclusion that can be drawn from this example is that,
especially in panel block caving, the estimates of in situ ore reserves
are subsequently modified not only by the mine layout but also by the
draw practice: dilution is related to the evolution of the extraction.
Because of this increase in dilution as the extraction progresses,
normally the cutoff grade will be reached before 100% of the in situ
ore has been drawn, and the extraction will be stopped.   Therefore,
the recovery of in situ ore is not going to be 100%, and the last
tonnage to be drawn is going to be considerably more diluted than the
early tonnage.   Establishing the evolution of the dilution with the

extraction will determine the ore recovery, the overall dilution, and the evolution of the diluted grades of the ore throughout the life of a slice or panel; this information is going to be essential for production scheduling and blending.

As it is probably obvious from the previous pages, draw control and dilution control are the two main aspects of the grade control practice in a block caving operation.
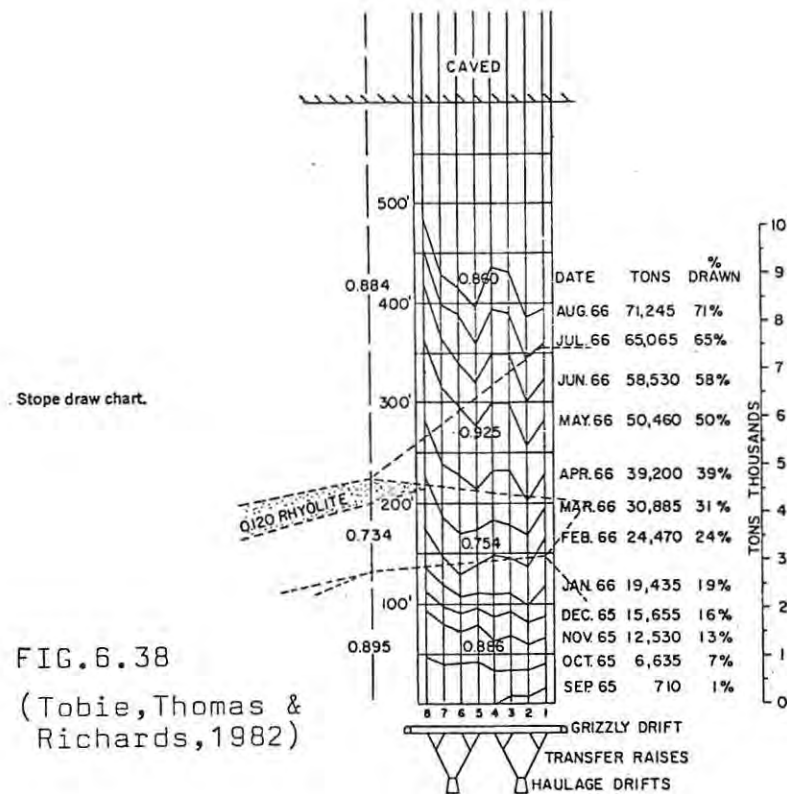
The main tool in the enforcement of grade control are the "draw charts", made based on the assumption that the ore and waste travel on vertical lines.

In the application of panel caving at Climax porphyry copper-molybdenum mine (Grould, 1982), the "angle of retreat" (see previous section) is the main factor affecting the grade control. Steep angles of retreat in panel caving maximize dilution. This is not only true at an overall scale but also at the small scale of two consecutive slices. At Climax the draw is controlled so that adjacent columns of ore are pulled to maintain no more than a 10% difference between them in total draw. This results in a maximum of 10% draw next to uncaved ore and 90 to 100% draw at the exhaustion line, with an overall ore-waste line lying at approximately 45° ("angle of retreat"), as finger spacing is approximately 10 m and the ore column is 91 m high; the active production area should therefore be maintained at a length of about 91 m. The ore grade at the draw points is monitored by grab samples taken by the hang-up men.

The angle of retreat can be modified to avoid drawing waste when the upper limit of the orebody changes in dip or elevation, or to relieve pressure on the production level openings or to cut down the rate of production, but always trying to maintain a less than 10% difference in total draw between adjacent columns. In a multi-production level block caving (s.s.) situation, in spite of the horizontal interface solid-caved material maintained throughout the mining of the block, a moderate retreat angle can be given to the final draw from the block,

so as to avoid some patches of waste that could exist in places at
the top of the block. (This is frequent whith blocks located at
the borders of the orebody, especially at the upper-outer borders
of irregularly-shaped bodies).

At San Manuel porphyry copper mine (Tobie, Thomas and Richards,
1982) panel caving is used and the draw at each draw point follows a
pre-established schedule. The ore drawn from each line of draw
raises is plotted by the tonnage department once a month, as shown
in Fig. 6.38. In these charts the extraction can be compared with

. Stope draw chart.

DATE	TONS	% DRAWN
AUG. 66	71,245	71%
JUL. 66	65,065	65%
JUN. 66	58,530	58%
MAY. 66	50,460	50%
APR. 66	39,200	39%
MAR.66	30,885	31%
FEB. 66	24,470	24%
JAN. 66	19,435	19%
DEC. 65	15,655	16%
NOV. 65	12,530	13%
OCT.65	6,635	7%
SEP. 65	710	1%

FIG.6.38

(Tobie,Thomas &
Richards,1982)

the in situ grades that are also plotted on them (as well as the main
geological/grade units). These charts are used by the block engineers
in predicting the grade of the draw (from the geological plus assay
information recorded on them), and in maintaining the desired angle of
draw over the individual grizzly lines. Adjacent charts can be

combined to optimize the angle of retreat in a direction perpendicular
to that of the individual charts.

At Henderson porphyry copper mine (Doepken, 1982), the ore draw is
controlled by assigning the number of LHD buckets to be drawn from each
draw point every day. Grab samples are normally taken from draw points
every 907 tons of production to establish the operating grade, and every
91 ton intervals when the extraction gets close to the cutoff grade.
The rate of draw is adjusted to the 0.3 m x day rate of cave, to
minimize the hazards of large voids or packing in the cave.   To
minimize dilution, the draw is scheduled to maintain an approximate
56° ore-waste contact (angle of retreat) between uncaved ore and
exhausted drawpoints.   This profile is maintained by progressively
increasing the total ore column draw in 15% increments in each row of
draw points, working away from the cave line.

Some typical total ore recoveries and dilutions, in different
mines using block caving, are shown in Fig. 6.39.

Ore Recovery and Dilution*

Mine	% Ore Recovered	% Dilution
Cornwall	100	17
Grace	85	20
Mather	67	10
Thetford	100	20
Creighton	95	15
Climax	92.5	15
San Manuel	101.5†	12.2
Urad	101†	15
De Beers	100	20
Ei Teniente	90.95	10–20

* Source: *SME Mining Engineering Handbook*, 1973.
† Actual recovery of calculated ore.

FIG.6.39   (Tobie & Julin,1982)

Costs and productivity

Tobie and Julin (1982) estimate the capital cost of mine, crushing
and milling facilities in $15 000 per short ton produced per day, basing
their estimate on 1976 costs;   apparently this capital cost would include

the mine developments. They also indicate that the mine operating costs, exclusive of development costs, should average $1.50 to $2.00 per short ton. Howard-Goldsmith (Min. Mag., Feb., 1978, p.111), reviewing the copper industry, quotes an overall cost per short ton of $1.60 for block caving mining.

Caving is best suited for high rates of production, what makes this method very useful to mine low-grade and high-tonnage-reserve mines. The range of daily production is shown in Fig. 6.40.

Daily Production Rates*

	Daily Production, t
Cornwall No. 3	1814
Cornwall No. 4	3629
Grace	6350
Mather	7676
San Manuel	58 967
Creighton	12 700
Thetford	3629
El Salvador	23 587
Climax	42 638
Urad	5715
Hecla's Lake Shore mine, oxide	5851
sulfide	9979
Atlas Consolidated	19 958
Bultfontein	2550
El Teniente	36 287

Source: *SME Mining Engineering Handbook*, 1973.

FIG.6.40   (Tobie & Julin,1982)

The success of block caving is dependent on a fairly high productivity. Production crews should be expected to produce in the range of 136 to 317 tons per man-shift, whereas productivity for the entire mine personnel, including service departments and management, ranges from 18 to 36 tons per man-shift (Tobie and Julin, 1982). A productivity of 192 wet tons per man-shift for production crews, and 24 dry tons per man-shift for the entire mine personnel, has been the average at El Teniente (Sisselman, 1978) while mining mostly "non-LHD" block caving; the data is shown in Fig. 6.41.

El Teniente factsheet

```
Work force ........................3,800 (including Future mine)
Productivity ........................24 dry mt per man-shift
Miner (buitrero) output ..............192 wet mt per man-shift
Number of blocks in production .......30
Production surface .................143,000 sq m
Extraction speed ...................0.39 dry mtpd per sq m caved
Drilling factor .....................3.00 m per sq m caved
Blasting factor ....................3.40 kg per sq m caved
Explosives consumed .............34 g per dry mt caved
Timber consumed .................0.177 ft per dry mt
```

FIG.6.41   (Sisselman,1978)

6.2. Sublevel caving

Sublevel caving was originally a variety of top-slicing applied in ground so weak that it would collapse, even in small openings, when the support was removed.   Heavily timbered drifts were driven across the orebody; the timber was removed at the end of the drift, and the ore caved and slushed out.   When the dilution was excessive the next set was removed and so on.   This method was slow, gave high dilutions and poor ore recoveries, and was labour-intensive.

In more recent times the method has been adapted to stronger ground which needs to be drilled and blasted; only the hangingwall waste caves in a natural way.   This method has been developed mainly in Sweden, particularly in the Kiirunavaara iron ore mine.   It has been used in base metal and asbestos deposits as well.

For a successful application of sublevel caving, the ore should be strong enough to stand without excessive support (because of the large amount of development work), to provide reasonably strong brows (the juncture of the blasted fan of drill holes with the back of the drift), to be suitable for drilling up holes 15 m or more in length, and for the drill holes to stay open for loading with explosives. Since the broken ore is surrounded on three sides by broken waste, strong brows and good fragmentation are necessary for the control of the dilution.   The waste should be weak enough to cave.

A vertical dip of the orebodies is the most favourable to the application of the method, and it permits the working of orebodies as narrow as 4 m (if the sidewalls are weak enough to cave with such a small opening). With flat dipping orebodies (see Fig. 6.58), large widths are necessary, and dilution increases towards the hangingwall and ore recovery decreases particularly towards the footwall contact.

The advantages of sublevel caving include: (i) it can be applied to both hard and moderately weak ground; (ii) it is flexible so it can be applied to irregular orebodies and wide or narrow orebodies down to about 4 m; (iii) all operations take place in drift-size headings that can be well-supported, providing safe working conditions; (iv) it is suitable for a high degree of mechanization; (v) activities can be specialized, simplifying the training of personnel and reducing the number of miners required; (vi) no pillars are left for subsequent high cost mining or lost ore; and (vii) the method has been successfully applied to pillar recovery. The major disadvantages are the high dilution and ore losses, and the problem of controlling them, and the high cost of development.

If the ore is wide enough, steep enough, and will cave, block caving would be selected as the cost is normally lower than that for sublevel caving. However, sublevel caving could be a better choice if the ore was too hard to cave readily, if the dilution from the hangingwall was excessive due to a flat dip, or if the orebody was too narrow or irregular in shape for the bulk mining associated with block caving.

The mechanics of sublevel caving

The progressive failure of the hangingwall of an inclined orebody is critical to the operation of a sublevel caving system. The caved waste forces the blasted ore into the drawpoints in the sublevels, making the loading of the ore easier and safer.

Fig. 6.42 shows that either starting the sublevel caving operation from the surface or from the bottom of an open pit (that mined the upper portion of the orebody), failure of the hangingwall follows a very similar

behaviour pattern. Cavability of the hangingwall of a sublevel
caving operation has got many elements of the slope failure analysis
in an open pit, as well as a few elements of hangingwall caving in
block caving.

(a) Mining from outcrop

(b) Failure of overhanging wedge

(c) Formation of steep face

(d) Development of tension crack and failure surface

(e) Development of second tension crack and failure surface

(f) Initial open-pit

(g) Formation of tension crack and failure surface

(h) Second tension crack and failure surface

(i) Progressive failure with increase in mining depth

FIG.6.42
(Hoek,1974)

Progressive failure sequence with increasing mining depth

Hoek (1974) has presented a method of analysis of a progressive
failure sequence with increasing depth of mining, based on the idealized
model shown in Fig. 6.43.   For the hangingwall it is assumed that there
are no dominant structural weakness planes (apart from those dipping
parallel to the orebody), and that the failure surface is free to find
its own path of least resistance through the rock mass.   The influence
of groundwater is not included in the analysis, since it results in
an appreciable increase in the complexity of the analysis and the drainage
provided by the underground workings tend to reduce its effects.
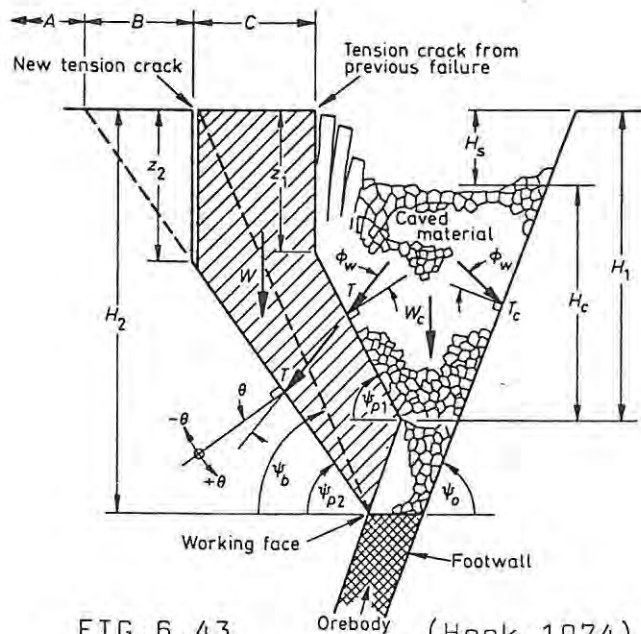
FIG.6.43   Orebody   (Hoek,1974)

Theoretical model for analysis of failure induced by caving. Stability regions: A. stable, suitable for permanent structures if H_2 is final mining depth; B. partially stable, unsuitable for structures sensitive to differential movement; C. unstable, prohibited access

Rock mass type	Angle of friction, ϕ, degrees	Cohesive strength lb/ft²	kg/m²
Soft rock, deeply weathered, with clay gouge in shear zones	25–30	4 000	20 000
Medium to hard rock, tectonically disturbed with close joint spacing	30–35	10 000	50 000
Hard rock with interlocking angular particles and very little clay mineral content	40	16 000	80 000

FIG.6.44   (Hoek,1974)

The downwards forces due to the weight of the wedge and the caved material will cause failure along the plane inclined at ψ_{p2} when these forces exceed the shear strength of the rock mass in the direction ψ_{p2} . This shear strength is made up of a friction component ϕ and an effective cohesive strength c (see table in Fig. 6.44).

Hoek (1974) presented the charts in Figs. 6.45 and 6.46 to estimate the angle of break (ψ_b) and the failure plane angle (ψ_{p2}) as functions of the density of the undisturbed rock mass (γ), of its effective cohesive strength (c), of the depth (H) and the depth of the caved material (Hc), and of the friction angle within the undisturbed rock (ϕ). The knowledge of the most probable angle of break is vital if surface installations, waste dumps and tailings dams are to be located safely on the hangingwall.

Dimension *H* can be either H_1 or H_2, depending upon which of these can be estimated on site, provided that H_c is measured from the same point

FIG.6.45
(Hoek,1974)

Failure plane angle chart
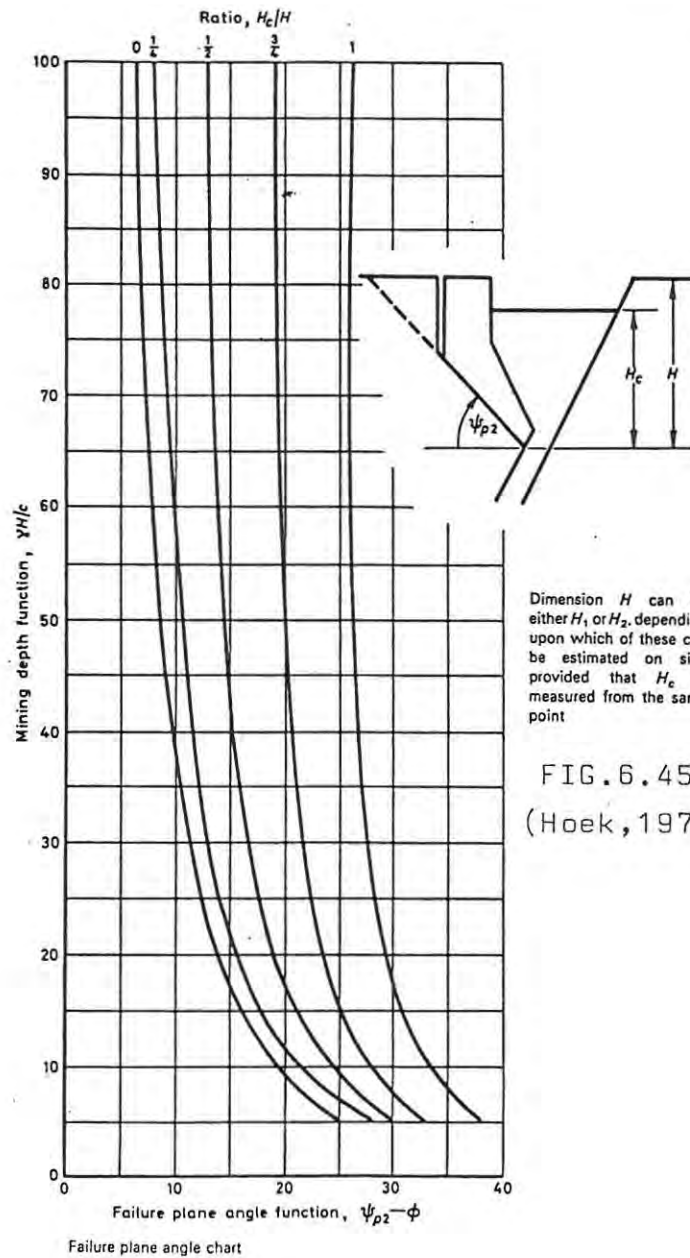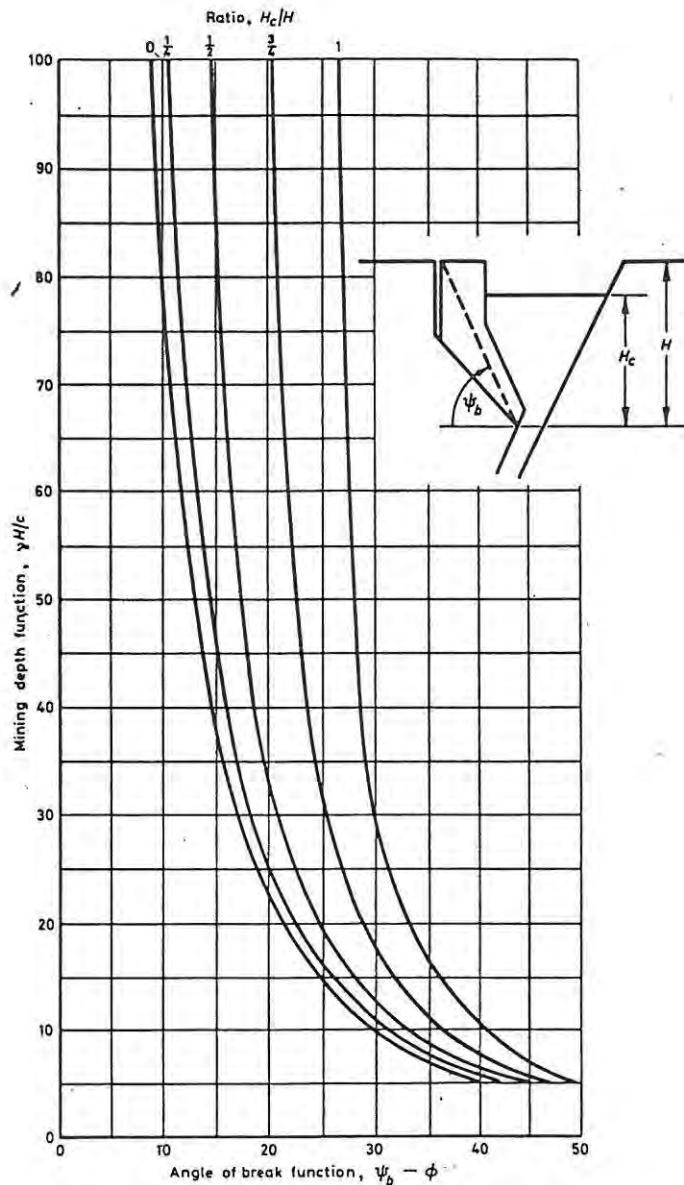
Angle of break chart FIG.6.46 (Hoek,1974)

Another important aspect of Hoek's work (especially to the operation of a sublevel caving system) is the prediction of the depth H_2 at which a new failure of the hangingwall will occur:

Mining depth for new failure
The mining depth H_2 at which a new failure occurs is obtained by solving equation :

$$\frac{\gamma H_2}{c} = a + (a^2 + b)^{\frac{1}{2}}$$

where

$$a = \frac{\gamma z_2}{c} \cdot \frac{\mathrm{Tan}\psi_o}{(\mathrm{Tan}\psi_{p2} + \mathrm{Tan}\psi_o)}$$

$$b = \left[\left(\frac{\gamma H_1}{c}\right)^2 (\mathrm{Cot}\psi_{p1} + \mathrm{Cot}\psi_o) - \left(\frac{\gamma z_2}{c}\right)^2 \mathrm{Cot}\psi_{p2} \right.$$

$$\left. - \left(\frac{\gamma z_1}{c}\right)^2 \mathrm{Cot}\psi_{p1} + K \frac{\gamma_c}{\gamma} \left(\frac{\gamma H_c}{c}\right)^2 \frac{\mathrm{Sin}\,(\theta - \phi)}{\mathrm{Sin}\,(\psi_{p2} - \phi)} \right]$$

$$\frac{1}{(\mathrm{Cot}\psi_{p2} + \mathrm{Cot}\psi_o)}$$

Input data required
H_1 Initial mining depth at which caving occurs
ψ_{p1} Initial failure angle or pit slope angle
z_1 Initial tension crack depth
γ Density of undisturbed rock mass
γ_c Density of caved material
ψ_o Dip of orebody
ϕ_w Friction angle between caved rock and undisturbed rock
ϕ Friction angle within undisturbed rock
c Effective cohesive strength of undisturbed rock mass
H_c Depth of caved material

Brown and Ferguson (1979) used Hoek's techniques to predict the progressive hangingwall caving at the Gath's chrysotile asbestos mine in Rhodesia; the results are reported to be very good.   The same analysis was applied to the footwall by substituting the adequate angles in the formulae (some of them becoming negative).

The second important aspect of the mechanics of sublevel caving is the gravity flow of the broken materials (ore and waste).   Its implications for mine design and control of dilution and ore recovery are discussed in other sections of this chapter.

Kvapil (1982) has demonstrated the mechanism of the gravity flow in broken material, on a very simple vertical glass model with horizontally layered white and black sand filling (see Fig. 6.47).   By connecting the individual black lines at the boundary of the volume of material affected by gravity motion, an elongated ellipsoid of revolution is obtained. After the extraction of a certain volume, the remaining material replaces this loss by loosening.   Although it is evident that a certain relation exists between the extracted material and the loosening, only the visible zone of the ellipsoid of loosening can be defined by this model.   The size of the ellipsoid of loosening increases as more material is extracted.

Mechanism of the gravity flow,
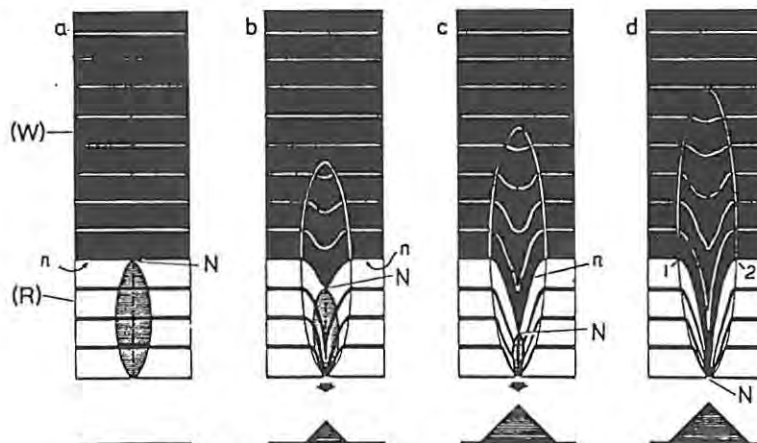
FIG.6.47   (Kvapil,1982)

The deflection of the lines of black sand fill at different stages of draw, gives a measure of the distribution of the velocity of flow in an ellipsoid of loosening. Zones of same particle velocity can be easily derived, and the analysis of these zones shows that not only the zone of loosening has the shape of an ellipsoid, but also does the zone from which the discharged material was extracted. This zone is called the ellipsoid of extraction.

Fig. 6.48 shows the effect that the material mobility (due to the extraction opening width, fragment size distribution, plasticity of the material, etc.,) has on the shape and eccentricity of the ellipsoids of extraction and loosening. Its most obvious consequence is the relative increase in dilution from overlying waste - and decrease in ore recovery - with smaller extraction opening widths, as seen in Fig. 6.49; exaggerated widths of the extraction opening produce mass flow in the center of the ellipsoid of extraction. Similar effects are observed in materials of low mobility (e.g. clay-rich materials or other materials that tend to pack).
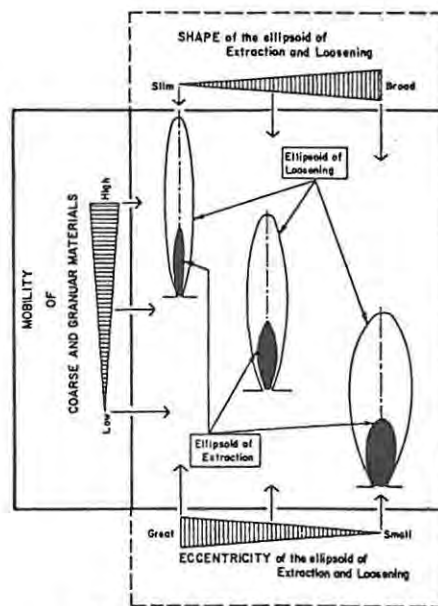
Shape and eccentricity of the ellipsoid of extraction and loosening as a function of material mobility.

FIG.6.48 (Kvapil,1982)
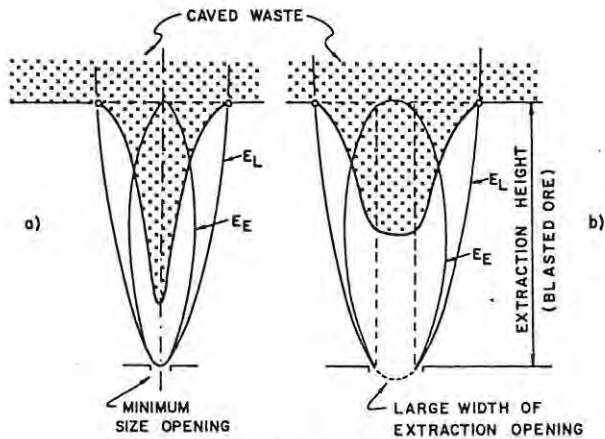
Influence of the extraction width on the gravity
flow pattern: a = small width and b = large width.

FIG.6.49   (Kvapil,1982)
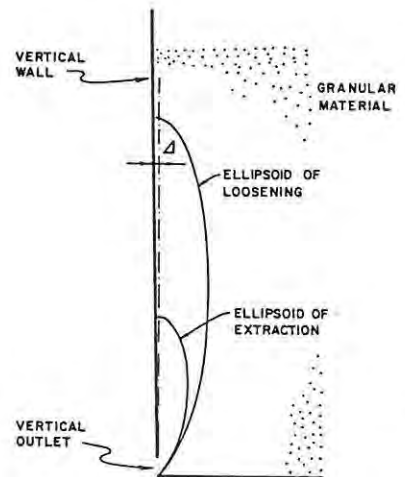
Scheme of ellipsoid of ex-
traction and loosening by extraction
through the opening located in verti-
cal wall. Vertical wall cuts off about
half of both ellipsoids.

FIG.6.50 (Kvapil,1982)

Fig. 6.50 shows the ellipsoids of extraction and loosening in
gravity flow by a discharge opening located in the vertical wall.
This situation is characteristic of the flow pattern in sublevel
caving in the direction of the production drifts.

Layout planning

Sublevel caving is a mining method that can result in high dilutions
and low ore recoveries if not applied properly.   A good layout design,
followed by the appropriate draw practice, is essential to minimize
dilution and maximize ore recoveries.   Furthermore, the high rate of
production developments per ton of ore mined (about 25% of the ore is
mined during the development stage), characteristic of sublevel caving,
stresses the need of good layout design.

The sublevel caving mining method consists in developing a haulage
and production drifts in sublevels, which are normally established at
7,5 to 12,5 m vertical intervals.   The haulage drift is driven in
waste, parallel to the strike of the orebody, and from it production
drifts cross-cut towards the orebody at roughly 10,5 m horizontal
intervals (see Fig. 6.51).   Slot raises are driven at the end of the

production drifts, near the ore-waste contact, and expanded to the
shape of the fan drilling.   Rings of up holes are drilled to break
towards the slot at burdens of 1,2 to 1,8 m;   they are drilled past
the next sublevel to the production drift immediately above.
Initially the hangingwall may not cave and the operation will be
similar to sublevel stoping-open stoping, but in later stages it
will cave and force the broken ore into the drawpoints at the
production drifts.
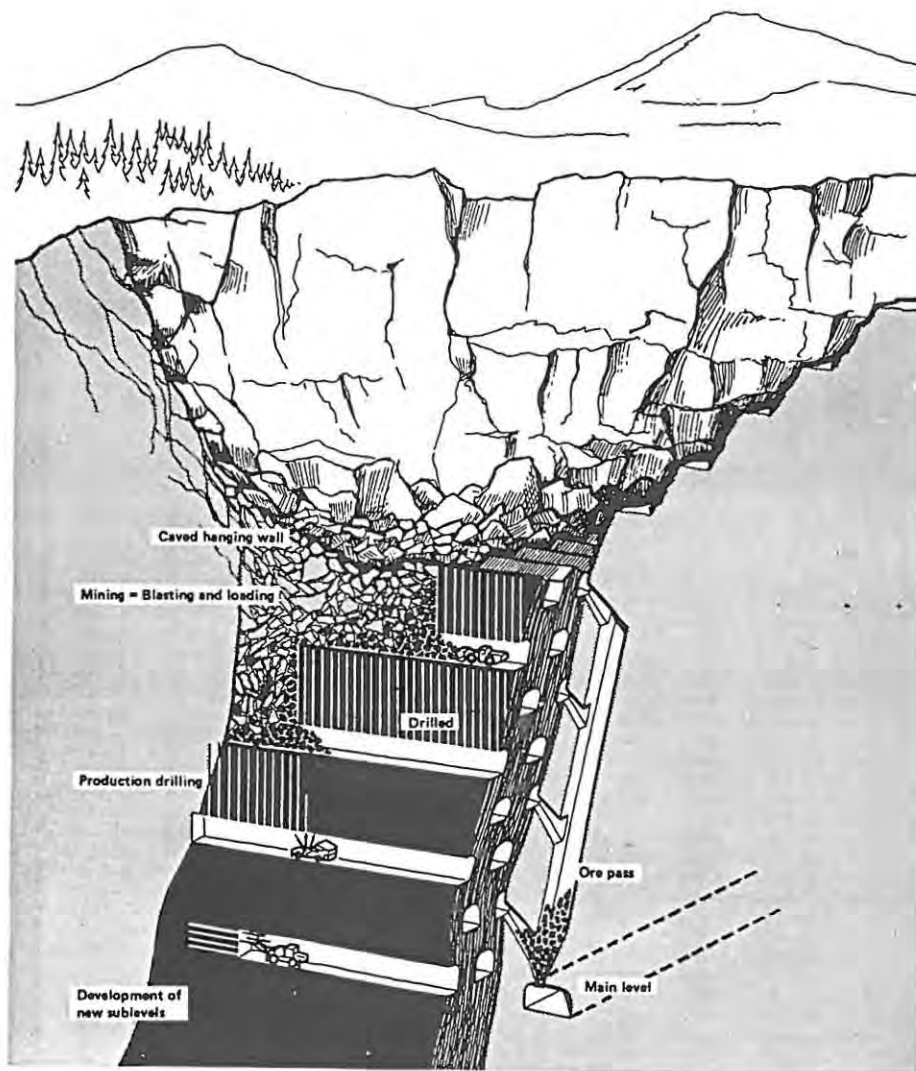
FIG.6.51   (Hamrin, 1982)

Sublevel caving in a large and steeply dipping ore body.

Transverse layouts are used in wide bodies; the production
drifts are normal to the strike of the orebody, or oriented in an
angle to the strike to accommodate the turn radius of the transport
equipment.   When the orebody widths are less than 12 to 15 m, the
amount of development per ton mined increases considerably, the
transverse layout becomes impractical and a longitudinal layout is
preferred.   In a longitudinal layout one or more production drifts
(according to the width of the orebody) are driven along the strike
of the orebodies.   Figs. 6.52 and 6.53 show the way transverse,
longitudinal and multilongitudinal layouts have been used at the
Granduc copper mine in Canada (Hancock and Mattson, 1982), in order
to obtain the optimum sublevel caving mining of the orebodies shown
there; the chalcopyrite ores occur in very irregular bodies of
disseminated to irregular semi-massive stringer mineralization,
varying in width from 3,6 m to over 18 m, dipping some 75° to the west
and extending over an 800 m north-south length.
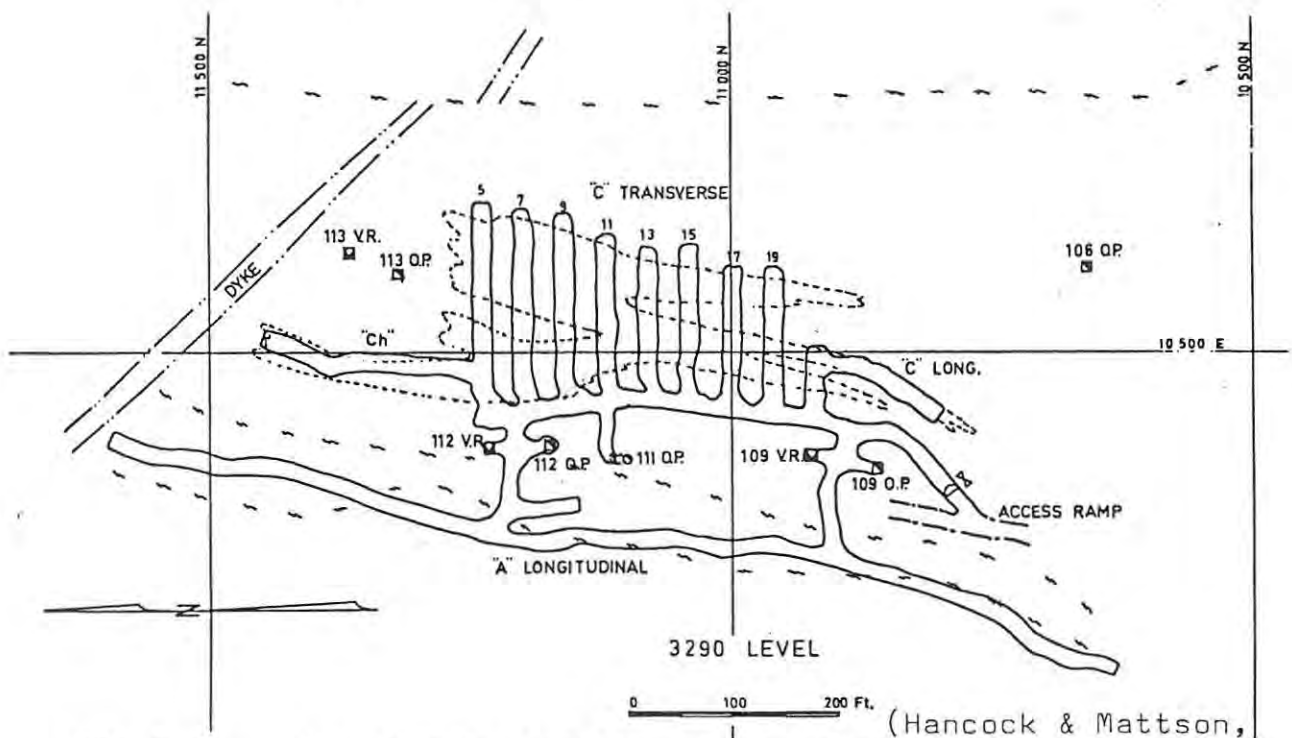
FIG.6.52   Typical sublevel plan showing transverse and longitudinal sublevel layout.   (Hancock & Mattson, 1982)

(Hancock & Mattson, 1982)
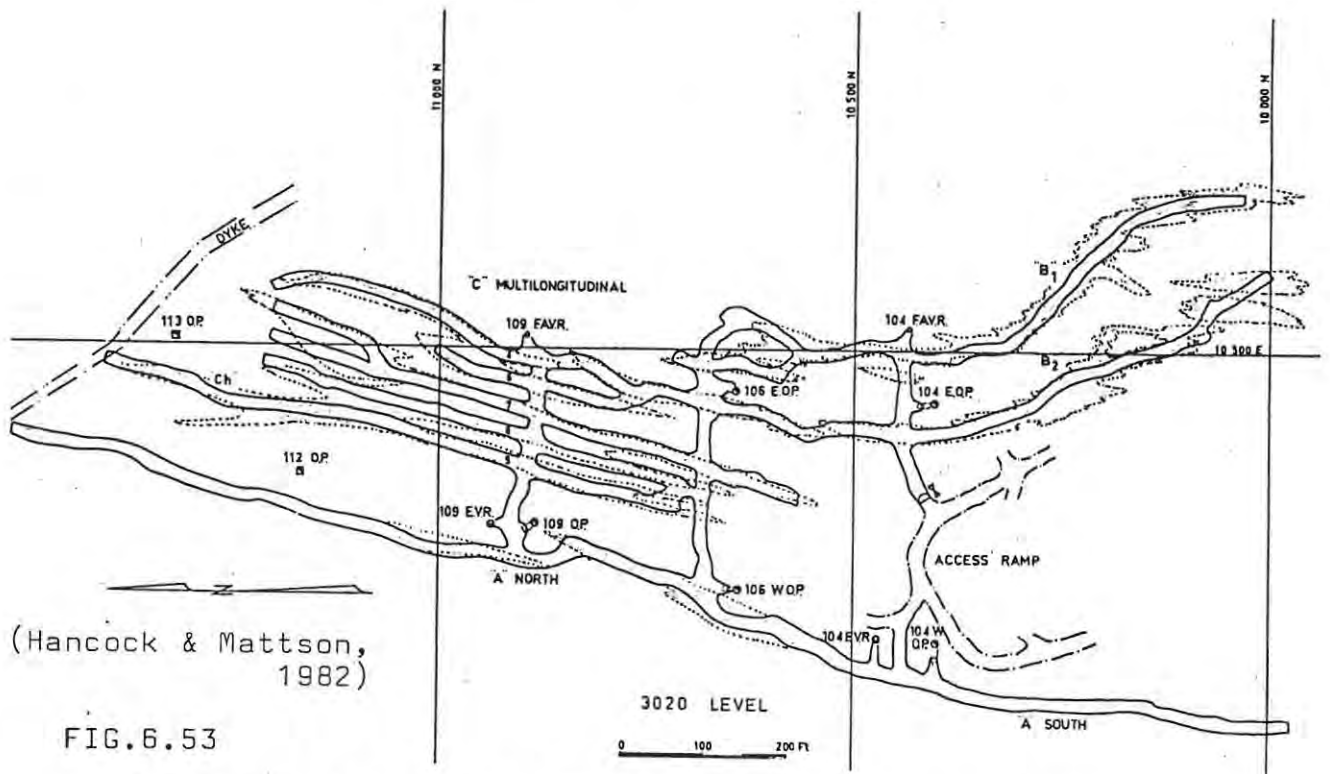
FIG.6.53

Typical sublevel plan showing longitudinal and multilongitudinal sublevel layout. Metric equivalent: ft $\times$ 0.3048 = m.

Recovery is normally better with a transverse layout as the production drifts cover all the ore from hangingwall to footwall, whereas the longitudinal drift tends to leave ore behind in an irregularly shaped orebody (see Fig. 6.54); in the last situation, it is difficult to adapt the blasting pattern in order to follow the irregularities. Because of the low ore recovery and the intense supervision demanded by longitudinal sublevel caving of irregular bodies, and because in these narrow bodies the necessary caving of the hangingwall is difficult to obtain, normally other alternative mining methods are preferred.
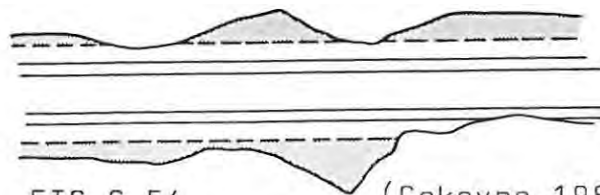
FIG.6.54 (Cokayne,1982)
Longitudinal layout. Hatched areas represent ore not recovered.
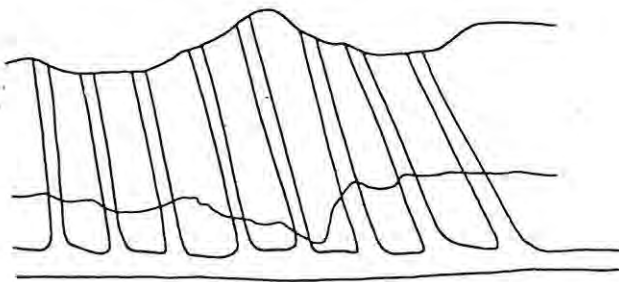
Transverse layout. All ore recovered.
FIG.6.55   (Cokayne,1982)
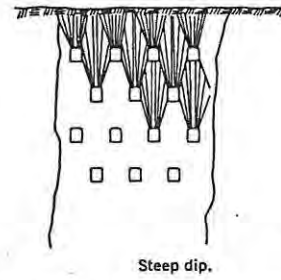
Steep dip.
FIG.6.56 (Cokayne,1982)
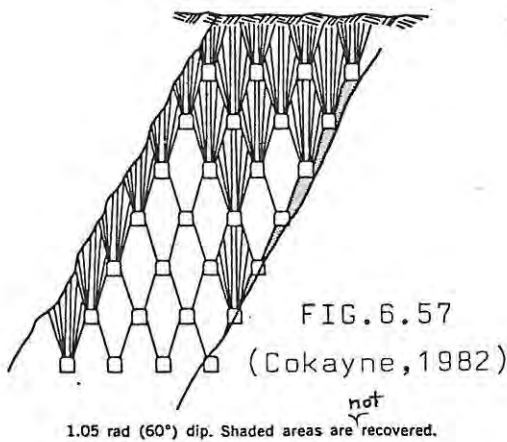
FIG.6.57
(Cokayne,1982)

1.05 rad (60°) dip. Shaded areas are not recovered.
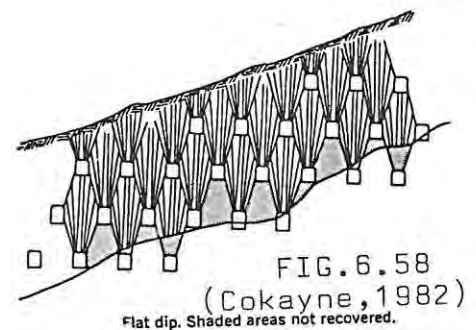
FIG.6.58
(Cokayne,1982)
Flat dip. Shaded areas not recovered.

For transverse layouts the haulage drift is driven in waste, preferably in the footwall, and 9 to 10 m from the ore contact to avoid ground support problems due to the blasting and caving. In longitudinal layouts, crosscuts to orepasses driven in the waste are situated at convenient intervals (determined by the rate of production and transport considerations).

The sublevel interval, or sublevel height, should be as large as possible - thus reducing the cost of development - , but in practice it is limited by several factors. If the dip (in a longitudinal layout) or the plunge (in a transverse layout) of the orebody is vertical there is no constraint, but as it flattens the sublevel height has to be reduced to avoid drawing the hangingwall waste and to increase the recovery of ore close to the footwall contact (see Figs. 6.56, 6.57 and

6.58). Other factors limiting the vertical distance between sublevels are: the increase in drilling costs, and the decrease in drilling accuracy due to hole deviation (resulting in poor fragmentation, and consequently in poor ore recoveries and high waste dilution) when the distance between sublevels becomes large. Common sublevel heights range from 9 to 11m, with maximum hole lengths of 15 to 19 m.

The maximum spacing between production drifts in a particular sublevel is dependent on the width of the draw ellipsoid, which, in turn, is determined by the extraction heading width, the height of the draw, and the eccentricity of the ellipsoid of draw (measure of the material mobility). The minimum spacing between production drifts should not be less than twice the extraction heading width, which ensures vertical mass flow of ore broken by parallel blastholes (see Fig. 6.59). The high amount of development per ton of ore for such a design, would be acceptable only in very competent rock allowing for wide production drifts, and when the increase in sublevel interval, due to the more simple parallel drilling of blastholes, compensates for the increase in developments per sublevel.

Values of extraction drift spacing used in different mines are generally in the range of 2.0 to 3.5 times the extraction drift widths. That spacing should be one that allows enough interaction between adjacent draw ellipsoids, reducing dilution and increasing recovery to an optimum when correlated with the increase in costs of developments. Being the maximum width of the extraction headings and height of the draw relatively fixed parameters, the eccentricity of the ellipsoid of draw is probably the main factor in the determination of the extraction drift spacing; Just (1972) suggests in situ tests to determine that eccentricity which would vary from 0.90 to 0.98, the most commonly accepted values lying in the range 0.92 to 0.96.
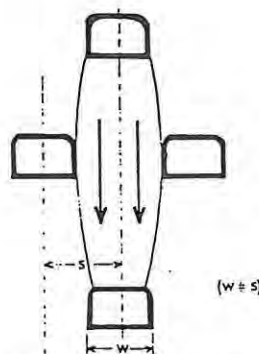
Parallel gravity flow occurring when extraction
heading width is extremely large

FIG.6.59 (Just,1972)
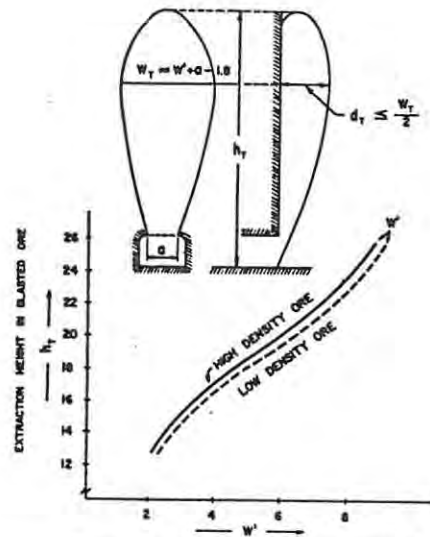
Very approximate width W' of the extraction ellip-
soid in high or low density ore as a function of extraction
height h_T.

FIG.6.60 (Kvapil,1982)

Kvapil (1982) has developed some practical guides to optimal
design.   Fig. 6.60 shows the theoretical width W' of the extraction
ellipsoid corresponding to the different total extraction heights
h_T , within the limits 15 to 26 m, for low and high density blasted
ore, and for a minimum extraction heading a = 1.8 m.   A very approximate
value of the total width W_T and total depth d_T of the extraction
ellipsoid, for a given extraction height h_T and opening a, can be
calculated in meters using the following empirical formulae:

$$W_T \approx W' + a - 1.8$$
$$d_T \leqslant W_T / 2$$

Knowing the sublevel height h_T and the total width W_T of the
extraction ellipsoid, it is possible to determine an approximate
horizontal spacing S_D of the extraction drift axes.   Assuming that
the relations and principles of idealized gravity flow described in
Fig. 6.47 can be applied to sublevel caving, and taking into consideration
that the total width W_T of the extraction ellipsoid is about 60 to 65%
of the width of the loosening ellipsoid (for heights of draw within the
range considered), Kvapil (1982) gives the following guides to estimate
the approximate horizontal spacing S_D of sublevel drifts:

(i) for extraction heights $h_s \leq 18$ m,

$$S_D < \frac{W_T}{0.6}$$

(ii) for extraction heights $h_s > 18$ m,

$$S_D < \frac{W_T}{0.65}$$

In conventional sublevel caving it can be added that $S_D \leq h_s$.
Fig. 6.6 shows the main elements of sublevel caving geometry. The
thickness of the blasted slice (burden spacing) on the front of the
sublevel is usually:

$$b \leq \frac{d_T}{2}$$

For heights of draw from 12 to 30 m, ring burdens of from 1.5 to
3.7 m have been used.

The sublevel front is usually inclined at 80°, making easier the
drilling and charging of the blastholes and maintaining a solid stable
brow, and minimizing the dilution. Fig. 6.62 shows that with an inclined
front the extraction ellipsoid is much slimmer than with a vertical front,
and it has a tendency to be inscribed within the blasted ore slice, thus
resulting in a smaller dilution.

Simplified principle of sublevel
caving geometry.
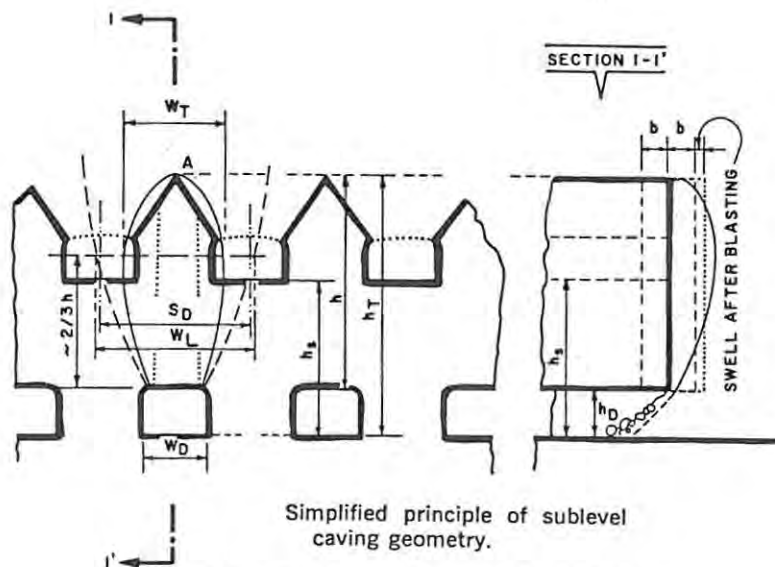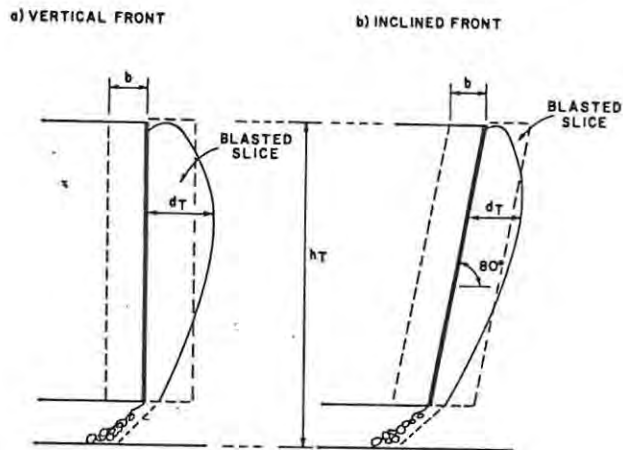
FIG.6.61   (Kvapil,1982)

Vertical (a) and inclined (b) front of sublevel caving. Inclined front increases the dilution.

FIG.6.62   (Kvapil,1982)
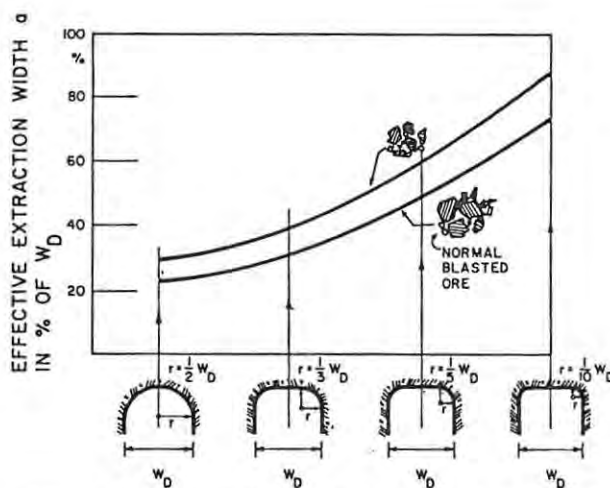
FIG.6.63   ROOF SHAPE   (Kvapil,1982)

. Approximate effective extraction width as the function of the shape of sublevel drift roof as a percentage of sublevel drift width W_D.

The size and shape of the production drift has an important bearing on the draw.   The drift should be as wide as possible, while still giving good support to the back and brows, and have flat hanging-walls; flat roofs (see Fig. 6.63) allow an even draw across the whole width of the drift, whereas with arched backs the draw will concentrate in the center and will not move on the sides (hence increasing dilution and reducing the ore recovery).   When ore is being extracted from a ring,

ore remnants do not flow and are left at the back of the muck pile,
out of range of the loader (see Fig. 6.64). The volume of ore lost
increases with increased ring burdens and drift heights, and decreases
when using loading equipment with high digging depth (e.g. LHD units
instead of overhead loaders on tracks). Single ring blasting reduces
dilution and is preferable to multi-ring blasting. From a grade control
point of view, the rings must not end in high grade ore, because when the
next round is blasted this ore is going to be lost in the back of the
muck pile; to avoid these situations, the burden can be modified or
multi-ring blasting adopted.

The drift height should be as low as possible and low profile
equipment should be selected for sub-level caving operations; about
3.2 m is not an uncommon height for production drifts : it is consistent
with the size of the drilling equipment and ventilation ducts.
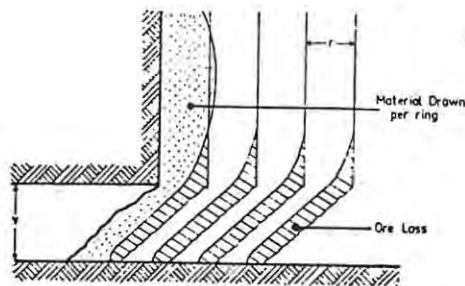
Sectional view showing ore pockets remaining
in an extraction heading after drawing ceases

FIG.6.64   (Just,1972)
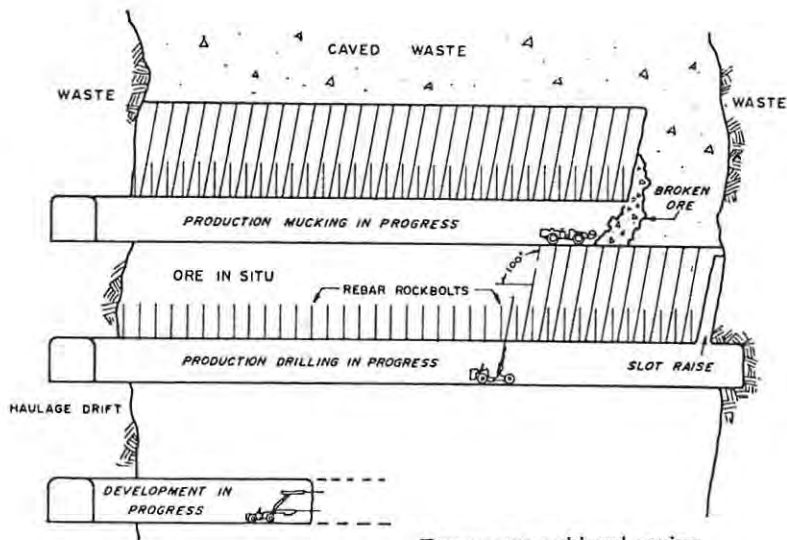
Transverse sublevel caving.

FIG.6.65   (Baase,Diment & Petrina,1982)

If the brows are strong and intact, recovery will be at a maximum and dilution will be minimized.   If the brow collapses, the ore will flood the drift and will cover the next row or rows of holes to be blasted; the holes will have to be dug out or they will be lost, and dilution will increase.   If the brow is uneven, the ore will funnel down through the high spots, reducing the width of the ore flow and resulting in ore losses.   A frequently used procedure to support the brows consists of systematically roof-bolting the backs of the drifts, as it is done at Craigmont copper mines in Canada (see Fig. 6.65).

The method of slotting in use at Craigmont is shown in Fig. 6.66. A slot drift (gives return ventilation as well) is driven along the ore-waste contact, and a slot raise is drilled at its end.   The slot raise is subsequently sliped open moving backwards along the slot drift, in the way shown in Fig. 6.66.   Fig. 6.67 shows the blasting pattern used at Craigmont during the production stage.   Note the unloded collars of the blastholes and the use of Shearex instead of 75% Forcite near them, trying to reduce the tendency to overload the collars (normal in fan patterns) and to protect the brow.   A characteristic of sublevel caving is that all blasting is against broken muck, which allows an expansion of about 15%; therefore, the powder factor needs to be approximately double that required for an open face, or from 0.3 to 0.4 kg of explosive per ton blasted.
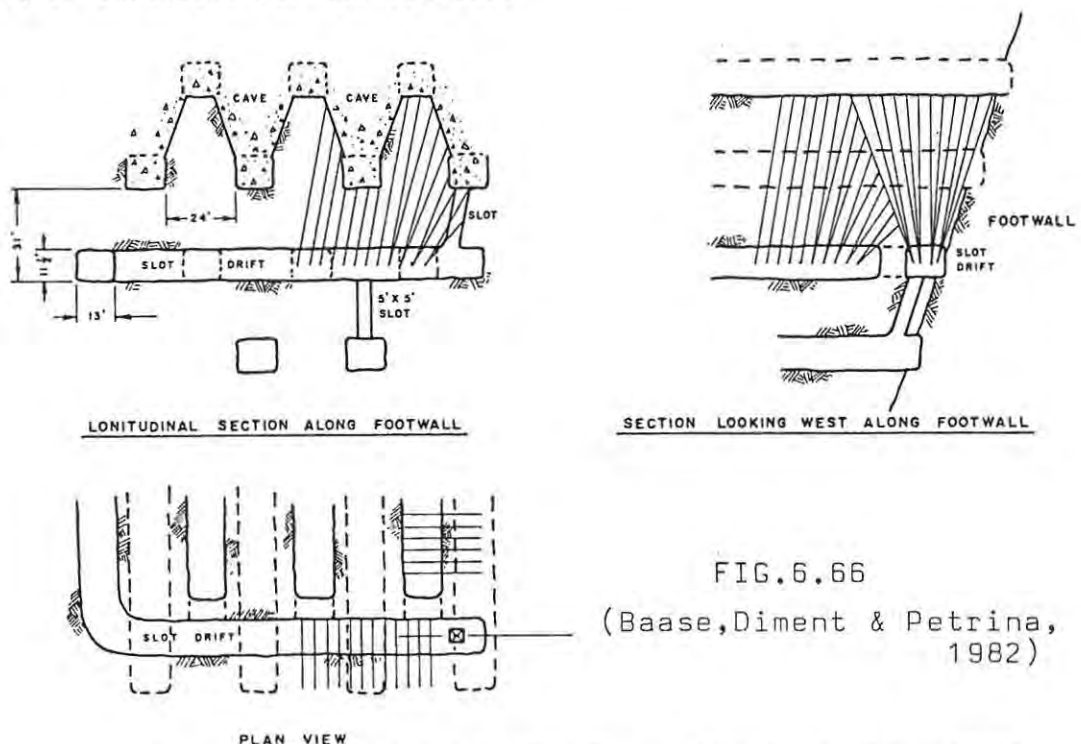
LONITUDINAL SECTION ALONG FOOTWALL

SECTION LOOKING WEST ALONG FOOTWALL

PLAN VIEW

Slot drift. Metric equivalent: ft $\times$ 0.3048 = m. (Original scale: 1 in. = 40 ft.)

FIG.6.66
(Baase,Diment & Petrina, 1982)

Fig. 6.68 shows a ring blasting pattern for a longitudinal sublevel caving layout.

A diagonal direction of retreat has been used at the Mufulira (see Fig. 6.69) and Kiirunavaara (see Fig. 6.70) mines, thus obtaining the best distribution of stresses over the production drifts. The stresses produced by the caving of the hangingwall are similar to those produced by block caving, whose effects on panel block caving were discussed in a previous section.

The sublevel shrinkage caving method shown in Fig. 6.71 is currently being tested at the Kiirunavaara and Konsul mines (Heden et al., 1982). A block consisting of seven sublevels is blasted, drawing only the excess volume of ore and leaving the rest behind. (The breaking method is the same as in ordinary sublevel caving). When the entire block has been blasted, all the broken ore is evenly drawn out downwards to a single loading level. The technique intends to reduce the dilution and improve the ore recovery.
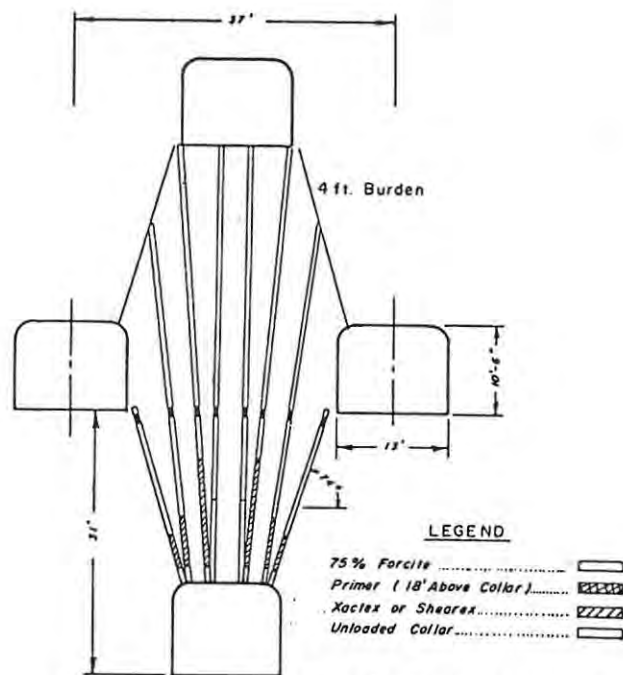
Current practice—single fan blast. Metric equivalents: 1 ft $\times$ 0.3048 = m; 1 in. $\times$ 25.4 = mm.

FIG.6.67 (Baase,Diment & Petrina, 1982)
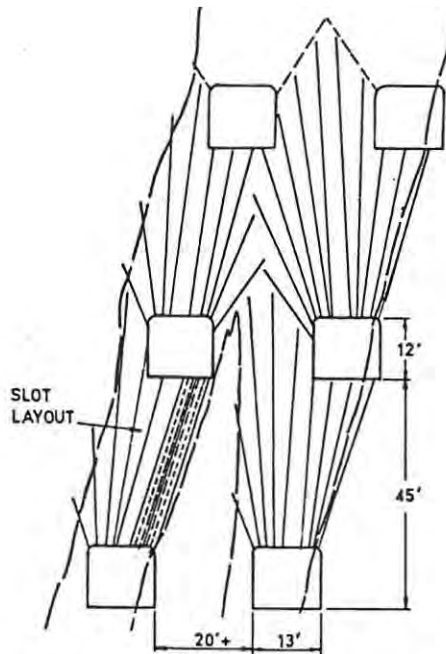
Cross section showing typical longitudinal sublevel and ring layout. Metric equivalent: ft × 0.3048 = m.
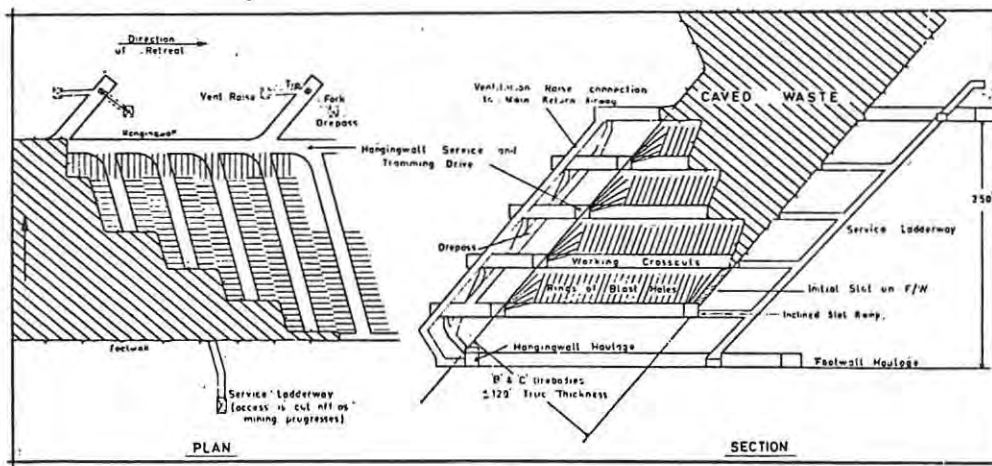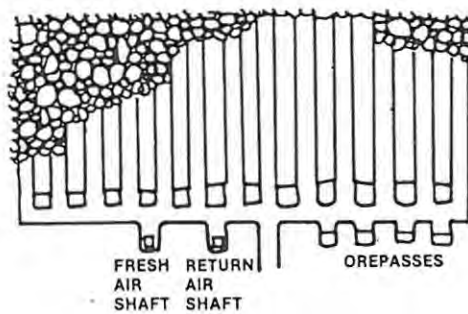
FIG.6.68 (Hancock & Mattson,1982)

PLAN

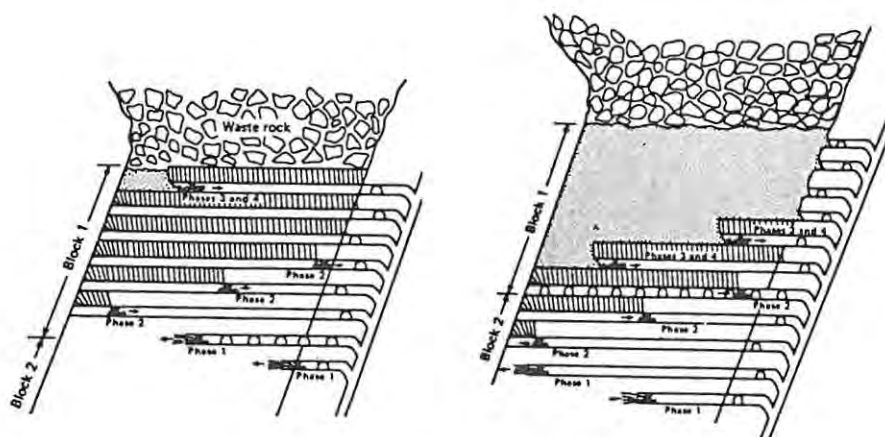Sub-level caving

SECTION

FIG.6.69 (Airey,1965)

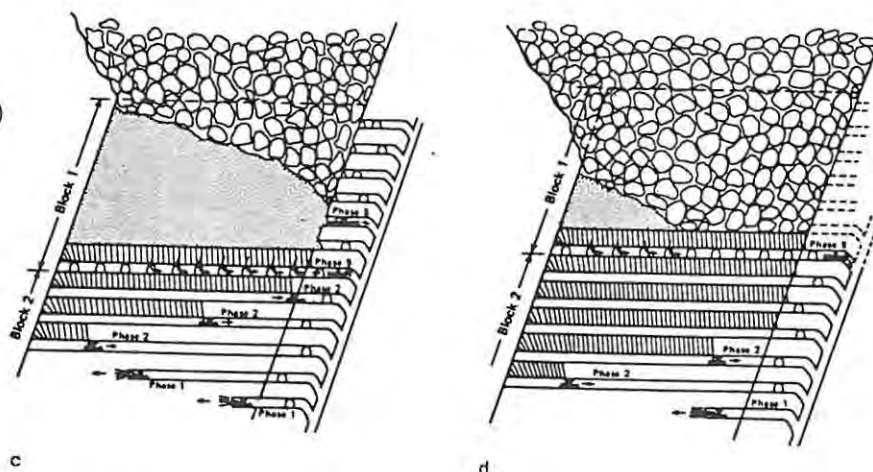

FRESH RETURN
AIR AIR   OREPASSES
SHAFT SHAFT

Caving technique.

FIG.6.70 (Heden,Lidin & Malmström,1982)

Sublevel shrinkage cav-
ing.

FIG.6.71
(Heden,Lidin &
Malmström,1982)

Evaluation and grade control

Sublevel plans, similar to the examples from Granduc mine shown in Figs. 6.52 and 6.53, are drawn once the sublevel height has been chosen.   The shape of the orebodies on these plans will determine the layouts to be adopted.

If a transverse layout similar to the one shown in Fig. 6.52 is adopted, then narrow or irregular short "tongues" of ore extending along the strike away from the main body (e.g. the one south of production drift 19 in Fig. 6.52) will not be recovered; they should not be included with the recoverable ore reserves.   Wider and longer "tongues" (see "Ch" and "C long" bodies in the same Fig. 6.52) can be recovered by locally using a longitudinal layout.

The pockets of waste in an orebody mined using a transverse layout can be blasted, partially drawn (just enough to account for the swell of the blasted material and avoid packing) and the rest left to mix with the caved waste.

In a longitudinal layout, small irregularities on the side-contacts of the orebody are not recovered (as shown in Fig. 6.54). When the orebody becomes wider, the additional ore can be recovered changing to a multi-longitudinal layout (see Fig. 6.53) and, in extreme situations, to a transverse layout.

The table in Fig. 6.72 shows a comparison of tonnage recovery, copper recovery and grade ratio, between transverse and longitudinal layouts at the Granduc copper mine (Hancock and Mattson, 1982).

Draw Statistics

	Trans-verse	Longi-tudinal	Combined
Tonnage recovery, %	129	132	131
Copper recovery, %	85	90	89
Grade ratio, %	67	68	68

$$\text{Tonnage recovery, } \% = \frac{\text{tons drawn} \times 100}{\text{in-situ tons blasted}}$$

$$\text{Copper recovery, } \% = \frac{\text{tons copper drawn} \times 100}{\text{in-situ tons copper blasted}}$$

$$\text{Grade ratio, } \% = \frac{\text{mill adjusted draw grade} \times 100}{\text{in-situ grade}}$$

FIG.6.72 (Hancock & Mattson,1982)

Once the mining layout has been designed, "in situ" (i.e. undiluted and unaffected by the draw practice) recoverable ore reserves can be estimated using the sectional method, working on the sublevel plans and on vertical cross-sections normal to the strike of the production drifts. The use of vertical cross-sections is essential to estimate the recoverable ore reserves in a longitudinal layout situation, as it can be seen in Fig. 6.68; when a longitudinal layout is being used, cross-sections are indispensable to mine planning, especially to design the ring blasting patterns.

A first estimate of the ore reserves can be obtained using the
data from surface diamond drilling, normally drilled on sections
perpendicular to the strike of the orebodies.   Once the orebody has
been developed, more accurate estimates can be made using data from
diamond drilling from the haulage drifts, channel or chip sampling
of the production drifts, and sampling of sludges from the fan-
drilling of blastholes.

At the Kiirumavaara iron ore mine (Lindstrom, 1971), "in situ"
recoverable ore reserves have been estimated using vertical cross-
sections perpendicular to the strike of the production drifts.
The estimates are broken down by ore qualities (according to the
iron and phosphorous contents), and according to the provenance of
the ore: from developments or from sublevel caving.   The grade and
tonnage of each round are estimated using the sampling results of all
four drifts involved in the round (see Fig. 6.73).   Each sample assay
result is taken to be representative of the ore in the immediate
vicinity of the drift (i.e. respectively contained in blocks A, B, C
and D, in Fig. 6.73), and the mean grade of the round (P_R) is calculated
as follows:

$$P_R = \frac{v_A \times P_A + v_B \times P_B + v_c \times (P_c + P_D)}{v_A + v_B + 2 \times v_c}$$

where V_A ... V_c is a weighting factor determined by the area of the
block around each drift contained in the caving round.   In extrapolating
the assay results in drifts B, C and D, allowance must also be made for
the inclination of the round.   This simple method has the advantage of
being easily processed by a computer.

The method can be adapted by (i) assigning geological areas of
influence to the samples taken in each drift, (ii) considering only the
limits of the recoverable ore, estimated superimposing the layout of the
ellipsoids of extraction, and (iii) using the data from the sampling of
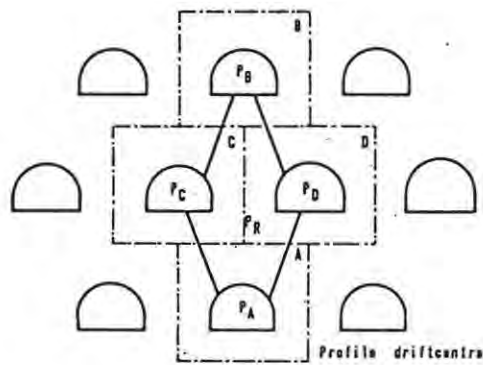sludges of the blastholes.

FIG.6.73
(Lindstrom,
1971)

— Calculation of average phosphorus content
in caving-blasts.

These estimates of "in situ" recoverable ore reserves will be subsequently modified by the dilution and ore losses derived from the blasting and draw practice.

The most common form of ore loss due to lack of accuracy in blasthole drilling is the formation of a solid bridge or arch, which is left behind (see Fig. 6.74). Blastholes that deviate from the planned ring pattern produce these bridges, and, in addition, result in coarse fragmentation that will allow the waste to dilute the ore. Other forms of dilution related to the blasting include the collapse of the brow and the channelling of the flow in uneven brows, both mentioned in the previous section.
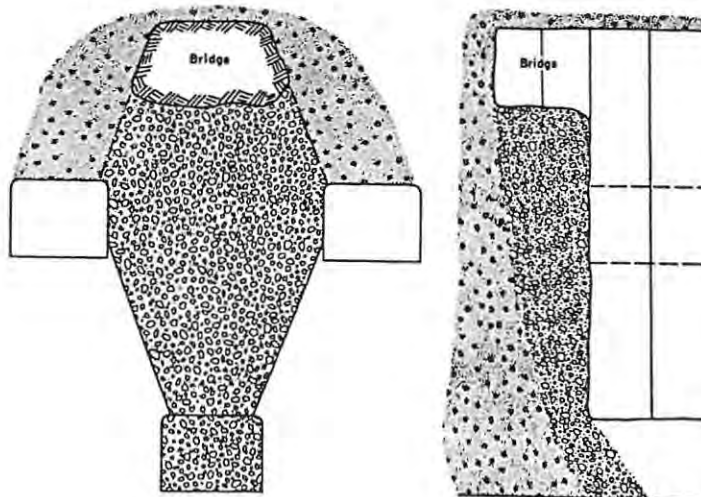
Formation of bridge.
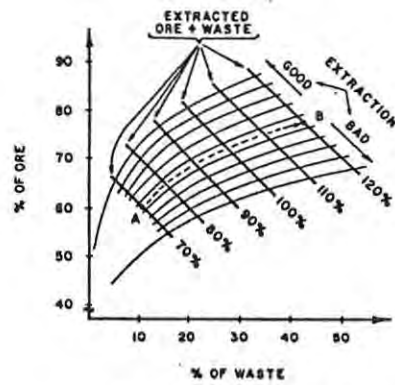
FIG.6.74
(Cokayne,1982)

During the process of drawing the broken ore, some is lost and dilution is introduced as a result of the flow pattern of the broken material discussed in previous sections. The dilution process in sublevel caving can take many different anomalous forms: outflow of waste at the beginning of the ore extraction due to arching in the broken material or to coarse fragments in it, random occurrence of waste rock pockets at different stages of ore extraction, etc.

The normal process of dilution development in sublevel caving is illustrated in Fig. 6.75. Theoretically the best extraction will be defined by the line OA, in which 100% of the volume of the "in situ" ore is extracted with no dilution. In practice, dilution will progress with the increase in volume of extraction (ore plus waste) following curves like I (reflecting a good draw control), II or III (reflecting a bad extraction or draw control); for example, after drawing 110% of the "in situ" ore (in volume) with a good draw control (curve I), 83% (in volume) of the "in situ" ore would have been recovered, and waste amounting to the equivalent of 27% (in volume) of that "in situ" ore would have been introduced as dilution. Fig. 6.76 shows a simplified chart of dilution development, for good to bad draw control.
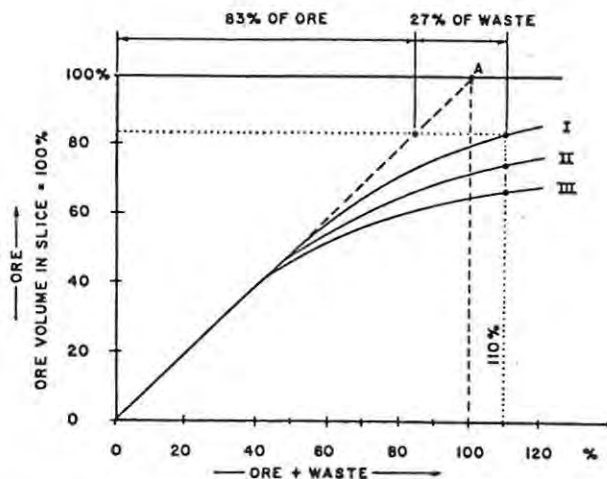
Generalized process of dilution development in sublevel caving. Curve I represents good extraction and Curve III represents bad extraction with high dilution.

FIG.6.75   (Kvapil,1982)

Simplified chart of dilution
development.
FIG.6.76 (Kvapil,1982)

By means of an economic analysis the cut-off grade is determined, thus defining the optimum extraction: the one at which the diluted grade of the ore extracted during the last extraction increment (one shift, etc.) reached the cut-off grade. The optimum extraction is not reached when the overall grade of the cummulative extraction is diluted to the cut-off grade; in this second situation, the last tonnages drawn before reaching this point could be well under the cut-off grade and its mining be uneconomic.

It is not easy to determine when the muck has reached the cut-off grade. The two general approaches to this subject are the visual method and the sample and assay system.

To use the visual method of draw control it must be possible to visually distinguish between the ore and the waste. Since it is difficult to accurately estimate the grade, the cut-off points are generally estimated based on the proportion of ore and waste in the muck; on the other hand, the draw does not reach the cut-off grade because the ore becomes poorer but because the proportion of waste dilution increases.

For example, if the in situ grade of the ore is 2% (estimated by sampling fan-drilling sludges, etc.), the desired cut-off is 0.5%, and the dilution carries no values, then mucking should cease when the muck pile appears to consist of 25% ore and 75% waste. The theoretical

number of LHD buckets to achieve this cut-off point is provided to the production supervisor, as well as the calculated desired percentage of ore and waste at this point. If the cut-off appears to occur before the theoretical number of buckets has been taken, the operator will take a fixed number more (30 buckets at the Granduc mine), if possible to a waste-pass, and stop the draw if conditions do not improve. If he completes the theoretical number and the ore-waste ratio is still above the cut-off one, he will continue to draw until reaching the cut-off point.

The sample and assay system is usually applied when it is difficult to distinguish between ore and waste. At the Granduc copper mine (Hancock and Mattson, 1982), 1.4 to 2.3 kg grab samples are taken from the muck pile after each new blast and at intervals of 20 to 30 scoop buckets. Whenever possible the visual method is preferred, since the sample and assay method has obvious disadvantages: (i) it is difficult to collect a representative sample from the muck pile, and (ii) the muck pile must be left idle until the assay results are obtained (and then the next muck pile may be significantly different from the ore sampled).

Fig. 6.77 shows an example of a draw control sheet, containing the instructions to the operators.

FIG.6.77   Example of a draw control sheet. (Hancock & Mattson, 1982)

Costs and productivity

Halls (1982) quotes an average cost of sublevel caving mining, in 1977 dollars, of $5.51 per ton, with $4.41 per ton being considered a low cost and $8.82 per ton a high one. Labour is said to constitute 64% of that total cost.

The graph in Fig. 6.78 shows the change in the cost distribution at the Kiirunavaara mine during the late 1960's, as a result of the introduction of LHD equipment. At the same time the productivity increased, as it can be seen in Fig. 6.79. At present, a productivity of 15 to 20 tons per manshift (total) is considered normal, and 36.6 tons per manshift (all underground personnel) has been attained.

Fig. 6.80 shows that when compared with other costs, capital costs have less influence at high capacities.

Cost distribution for main haulage level. (Courtesy B. Hedberg, IXth World Mining Congress, 1976.)
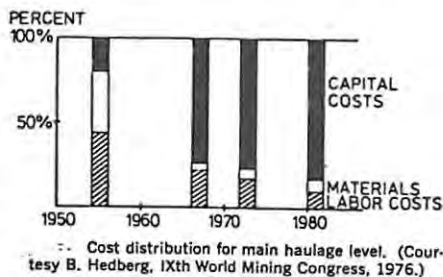
FIG.6.78 (Heden,Lidin & Malmström,1982)

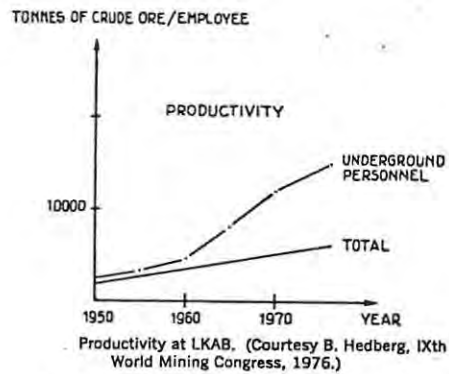Productivity at LKAB. (Courtesy B. Hedberg, IXth World Mining Congress, 1976.)

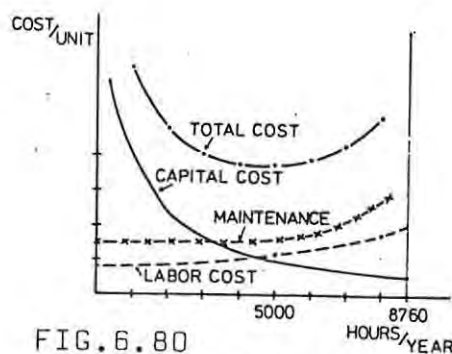FIG.6.79 (Heden,Lidin & Malmström,1982)

FIG.6.80

Cost relationships. (Courtesy B. Hedberg, IXth World Mining Congress, 1976.)

(Heden,Lidin & Malmström, 1982)

7.   SUBLEVEL STOPING

7.1. The application of sublevel stoping

The sublevel stoping mining method is usually applied to relatively steeply dipping, competent orebodies, surrounded by competent wall rock. Historically the method is believed to have originated in Canada, but it has been used worldwide, particularly in the mining of base metal deposits (veins, volcanogenic massive sulphides, etc.).

Sublevels, as well as an undercutting level with the cones to the drawpoints (see Fig. 7.1), are developed in ore above the footwall haulage level.   A vertical slot is cut at one extreme of the stope, and mining retreats away from that slot.   Vertical slices of ore are blasted against the slot, using rings of blastholes drilled from the sublevels.   The broken ore falls into the cones in the undercutting level and is drawn towards the footwall haulage.

FIG.7.1
(Thomas,
1978)

SECTION A-A   VERTICAL LONGITUDINAL PROJECTION
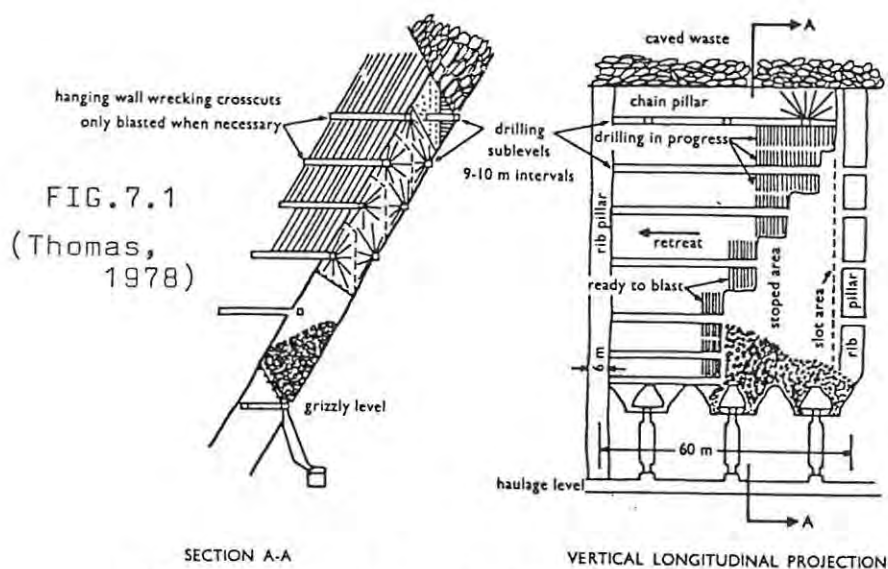
Open-stoping layout in steeply-dipping areas

The suitable orebodies must be steep enough to allow the broken
ore to flow by gravity down to the drawpoints. A successful method
has been developed (the "cascade" technique) for mining some orebodies
down to a dip of 25°, and scrapers have been used in some situations
where the stope footwall was less than the angle of repose.

Steeply dipping orebodies down to a width of about 1,5 m have
been mined using sublevel stoping. For widths less than about 6 m
the full utilization of long-hole drilling becomes difficult and
overbreaking inevitable, unless the orebodies are extremely regular
and with sharp contacts, or short holes and reduced sublevel vertical
intervals are used (with the resulting increase in development costs
and decrease in productivity).

There is no upper limit on orebody size : the geometric orientation
of the stopes is adjusted accordingly (transverse layouts for wide bodies),
and suitable support pillars are left. At the Strassa iron ore mine in
Sweden, a method to mine wide, steeply dipping, regular, competent orebodies
in competent country rocks has been developed.

A relatively uniform strike and a regular orebody outline is desirable,
since the possibilities of following irregular contours with long-hole
drilling and blasting are limited. Sharp contacts and ore that parts
easily from the sidewall waste are highly desirable. The hangingwall must
be competent enough to remain open while the ore is being blasted and drawn,
thus reducing the dilution.

With sublevel stoping the time of development is long and the capital
investment is extensive before stoping commences, but its production cost
per ton is low, dilution moderate, and safety conditions good, and the method
is normally a first choice for orebodies that meet the requirements.

7.2. Layout planning

The main elements of stope design in sublevel stoping are: (i) the
length and width of the stopes, (ii) the stope height, (iii) the distance
between sublevels, (iv) drawpoint location and design, (v) undercutting

(coning level), (vi) slot raising and slotting, and (vii) production drilling and blasting.

In sublevel stoping mining the stope must remain open while the ore is being blasted and drawn, and the dilution with waste material caved from the sidewalls or the hangingwall must be kept to a minimum. The whole concept of pressure arches discussed in the chapter on block caving also applies to sublevel stoping of steeply dipping orebodies.

Because of the arching above the open stope, the vertical pressures affect its sidewalls producing sets of shear fractures, similar to the ones shown in Fig. 7.2, and vertical tensional fractures that may produce

FIG.7.2
(Thomas, 1978)
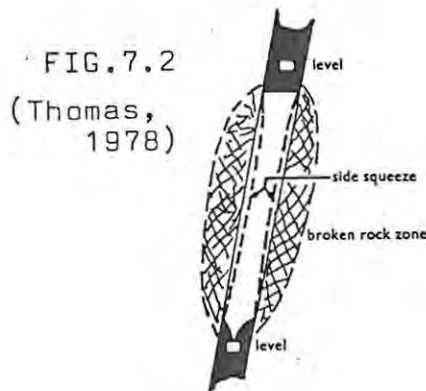
collapse of slabs of sidewall material (particularly in high stopes, and when vertical geological discontinuities in the sidewalls reduce their tensile strength). From Fig. 6.1 in chapter 6, it can be concluded that when arching takes place in the hangingwall (i.e. $\sigma_t = 0$) and it becomes stable, the stresses σ_c over the sidewalls increase to roughly 4 times the cover load (S_z). Considering the induced cleavages and abutment stress peaks of 4 to 5 times the cover load, and knowing the compressive strength of the rocks involved and the shear strength and orientation of geological discontinuities (faults, bedding, joints, veins, etc.) in the sidewalls, one can predict the depth and the stope dimensions that will result in rock mechanics problems.

In sublevel stoping the hangingwall of the stope is normally flat, and for small stope widths or spans (B) moderate compressive horizontal stresses (σ_t) will affect the roof rocks (see Fig. 6.1). But when the stope width increases, those stresses become tensional and the hangingwall will eventually cave. From Fig. 6.1 we can conclude that there is a maximum stope width (B), and that when that width is exceeded the hanging-wall is going to cave (due to the tensional stresses σ_t), and the sidewalls are going to collapse by slabing (due to increase in compressive stresses σ_c).

The rock mechanics analysis of these stresses is very complex, particularly because of the presence of geological discontinuities in the walls. In mines in production the problem is solved by testing different stope widths. In new mines experience from other mines, which worked under similar geological and rock mechanics constraints, is used. Maximum stable stope widths vary within a large range of values, from approximately 4 to 5 m to over 30 m.

The stope length is normally controlled by the intended production rate. Generally no more than about 3 rings of blastholes can be blasted at a time, and this is going to result in a fixed maximum production rate per stope (the other important factors are the stope height and the loading capacity per drawpoint). When the orebody is long enough and a larger production rate per haulage level is required, the length of the orebody can be divided in two or more stopes with rib pillars between them. Rib pillars are not indispensable (they normally protect the accesses to the sublevels), but a new set of travelling way-ore pass and slot raise is necessary; several alternatives can be used : for example, a central access (travelling way plus ore pass to dispose of the muck coming from the development of sublevels) and stoping retreating towards the center from slots in both extremes of the orebody.

Dividing the length of the orebody in two or more blocks has the advantage of making shorter the time of development, accelerating the start of production, and making more efficient the mucking in the interlevels.

The efficiency of the mucking in the sublevels is going to be the
other important limitation to the length of the stopes (unless more
than one set of ore-passes is used). When the mucking in the sublevels
is done with wheel-barrows, it becomes inefficient at lengths larger
than 30 to 40 m; at the other extreme, when using trackless loading
equipment in the sublevels (LHD units), the maximum efficient length
of the run (and of the stope) increases up to 100 or 120 m.

The stope height is controlled by the shape of the orebody, particularly
by its dip, width and height, by the competence of the stope pillar and
walls, by the slenderness ratio of adjacent pillars, and by the desire to
obtain a given production rate and optimum costs. The stope height, as
well as the stope width and the maximum number of rings that can be blasted
each time, are important factors affecting the production rate. This rate,
and consequently the stope height, is controlled and defined by the
maximum loading and transport rate that can be achieved in the haulage
level. On the other hand, the cost of haulage level developments and the
ore losses in bottom and crown pillars decrease as the stope height increases,
whereas the development time before production starts follows the inverse
relationship. The stope heights normally fall within the 40 m to 100 m
range, with 60 m (± 200 ft) being a frequently used height.

The distance between sublevels is controlled by the decrease in
accuracy of fan drilling (hole deviation results in coarse fragmentation
and over and underbreaking), the increase in drilling costs, and the lower
selectivity as distance between sublevels becomes larger. The limits of
the blasted area between two sublevels are fairly regular, and selective
mining of irregularly shaped orebodies involves a reduced distance between
sublevels (or, for example, the use of bench blasting instead of longhole
fan blasting). Distance between sublevels is generally in the range of
4 to 25 m, and 6 to 12 m is the most frequently used range.

The table in Fig. 7.3 shows some typical stope dimensions.

			Stope Dimensions, Ft				
Mine	Monthly Stope Prod., Tons	Vert. Haul. Interval Ft.	Height	Width	Vert. Sublev. Spac.	Hor. Sublev. Spac.	Ore Body
Holden, Wash. U.S.A.[77]	—	350	240	55	55–70	60	Pyritized shear zone in schist country rock. Length 2,500 ft, up to 80 ft wide, mined to 2,500 ft.
Noranda,[44] Canada	—	250	175	60	60	60	Massive sulfides in rhyolite and breccias. Elliptical in plan, 400 × 800 ft. Depth over 3,000 ft.
Mufulira, Zambia[72,95]	25,000 lt	250 250	210	35	30	35	Mineralized quartzites and graywackes. Three superimposed beds up to 45 ft thick; dip averages 45°.
Frood-Stobie, Canada[54]	20,000	350	308	70	70	70	Sulfide blebs contained in norite-quartz diorite. Lenses 9,600 ft long, 800 ft wide at surface, 4,500 ft deep.

FIG.7.3   (Haycocks,1973)

A good drawpoint system should have (i) an optimum spacing of drawpoints, within the constraints of stope dimensions, for uniform drawdown and maximum ore recovery, and (ii) a footwall haulage design for optimum loader (or transport equipment) maneuverability and ground stability in the bottom pillar.   The section of the raises to the drawpoints normally range from 1.2 m x 1.2 m to 2.2 m x 2.2 m, and on the level of undercutting ("coning level") they are sliped open ("coned") to 4 m x 4 m or 5 m x 5 m.   Therefore, 5 to 6 m between axis of draw-points is a frequently used distance.   When designing the drawpoint system, a plan of the area of the stope on the undercutting level is covered with circles with a diameter equal to that of the sliped cones (± 5 or 6 m), laid in a regular pattern that will achieve the desired uniform draw and maximum recovery.   The footwall haulage layout is designed according to this drawpoint configuration and to the loading and transport method in use (boxes/cars, LHD, overhead loaders/cars, scraper/cars, etc.).

To start the stoping, a vertical slot is blasted at the end of the stope opposite to that occupied by the accesses to the sublevels (travel-ing way plus ore pass).   During the stoping stage, mining retreats away from that slot, blasting towards this void.   The slot can be

started at a slot raise, raised by conventional methods, raise-
boring, drop-raising (predrilling and blasting a raise from the top,
using small diameter - less than 200 mm - holes for relief), or crater
blasting.   The slot is expanded to the full stope width, and should
be 4 to 5 m wide in the direction of stoping retreat.   Fig. 7.4 shows
two cross-sections through a stope at the Loraine Gold Mine (Bebb,
1982); the one on the left shows the pattern of production fan drilling,
while the one on the right is a cross-section through the slot, showing
the blasting pattern used to expand the slot raise to the full width of
the stope.   The method of undercutting in use at Loraine is shown in
Fig. 7.5.

Fig. 7.5 shows a method of undercutting compatible with the use of
overhead loaders or LHD units (provided that the right layout for foot-
wall crosscuts to the drawpoints is adopted).   Fig. 7.6 shows a method
of undercutting with long blastholes used at the Snow Lake gold mines in
Canada; the ore gravitates to a scraper drift, but the layout is compatible
with the use of loading boxes (chutes) and cars.   Normally undercutting
should not be carried out more than a few rings in advance of production
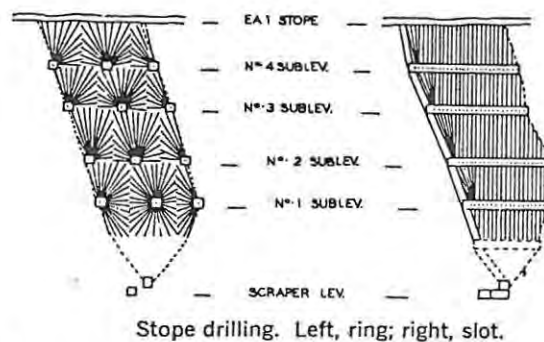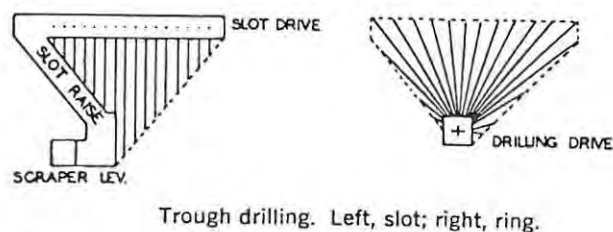blasting, mainly to ensure undercut back stability.

Stope drilling. Left, ring; right, slot.

FIG.7.4   (Bebb,1982)

Trough drilling. Left, slot; right, ring.

FIG.7.5   (Bebb,1982)

VERTICAL SECTION THROUGH UNDERCUT

A TYPICAL BLASTHOLE SLOT RING

PLAN VIEW OF UNDERCUT

A TYPICAL VERTICAL BLAST RING

Typical blasthole undercut drill ring layout.

SCALE IN FEET

0   10   20   30   40   50

LEGEND:

A. Winged Chute Raise.
B. Access Manway.
C. Scraper Drift
D. Draw Point.
E. Draw Point Footwall Wing Raise.
F. Draw Point Hanging wall Wing Raise.
G. Footwall Percussion Drilling Raise.
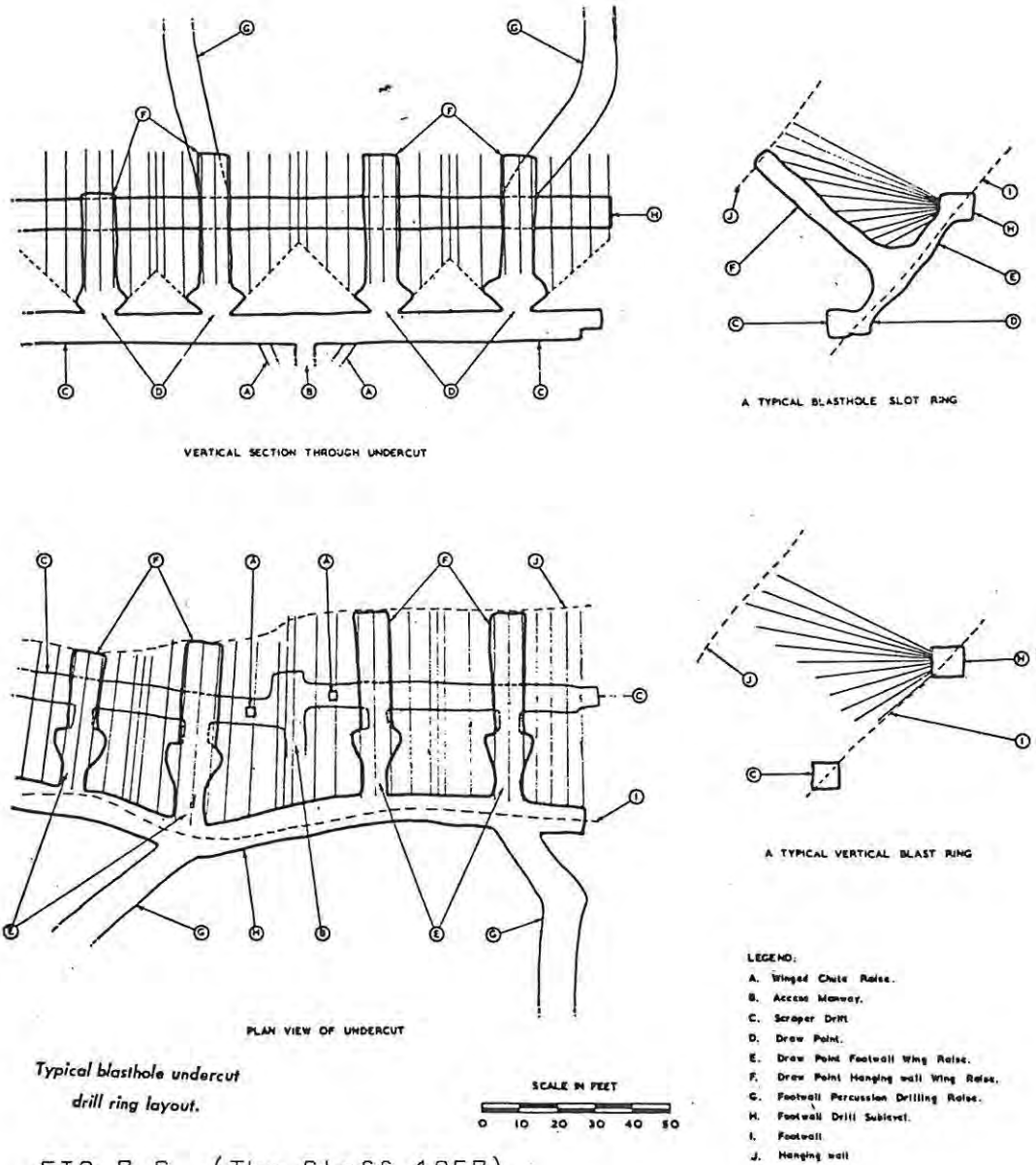H. Footwall Drill Sublevel.
I. Footwall
J. Hanging wall

FIG.7.6   (The Staff,1957)

Longitudinal layouts are used with narrow to moderately wide
steeply dipping orebodies.   Fig. 7.7 shows the developement layout
and the production blasting pattern in an approximately 20m wide
limestone orebody at the Bellafonte mine (USA);   an alternative
"one drift per sublevel" layout consists in the blasting of rings
of blastholes from production drifts in the center of the orebody,
but it generally results in a poor control of the overbreaking in
both sidewalls (thus increasing the dilution or, in case of underbreaking,
producing ore losses).
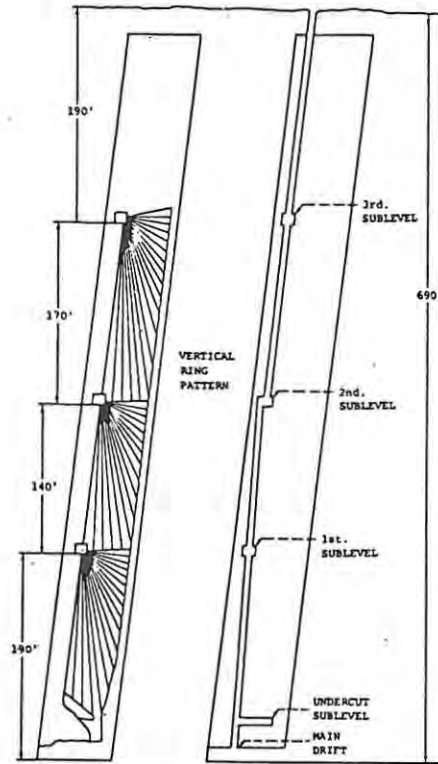
Cross sections, Bellefonte mine.

FIG.7.7 (Lawrence,1982)

Longitudinal layouts consisting of two (see Figs. 7.8, 7.9 and 7.10) or three (see Figs. 7.11 and 7.12) production drifts per sublevel are used with wider orebodies. The development of drifts along both contacts of the orebody gives a better control of the dilution by over-breaking and of ore losses. The fact that the blastholes along the sides of the stope are parallel to them, reduces the damage to the sidewalls and decreases the possibility of dilution by caving of waste. The development of two drifts per sublevel is generally uneconomic for stope widths of less than about 8 m.

In Figs. 7.8 and 7.11, the way the blasting pattern is adapted to follow the shape of the orebodies, and the range of possibilities of selectively mining orebodies of irregular shape, are noticeable; the selectivity of sublevel stoping is not as good as that of cut and fill or shrinkage. It is obvious that selectivity and grade control are highly dependent on the adoption of the correct ring-pattern design; sheets like the one shown in Fig. 7.13 have been used to transmit the instructions on ring-drilling pattern and blasthole loading to the operators.

FIG.7.8
(Lawrence,
1982)

FIG.7.9 (Beck & Staff,1957)
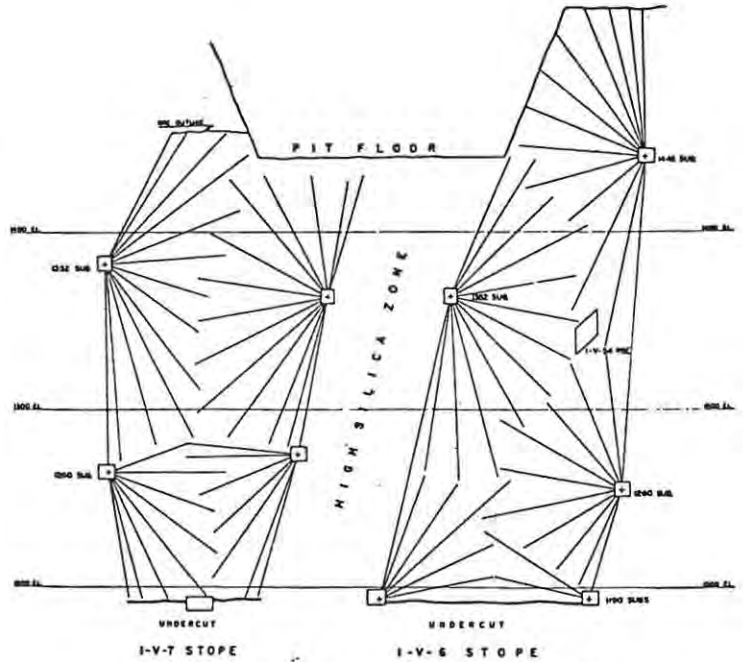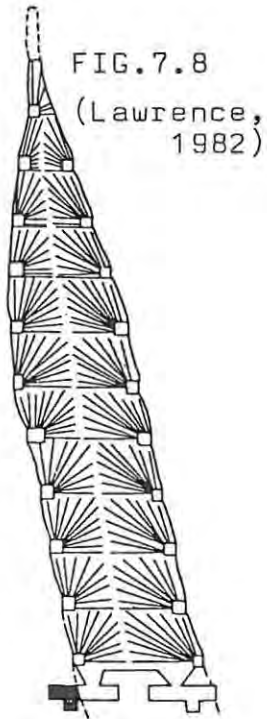
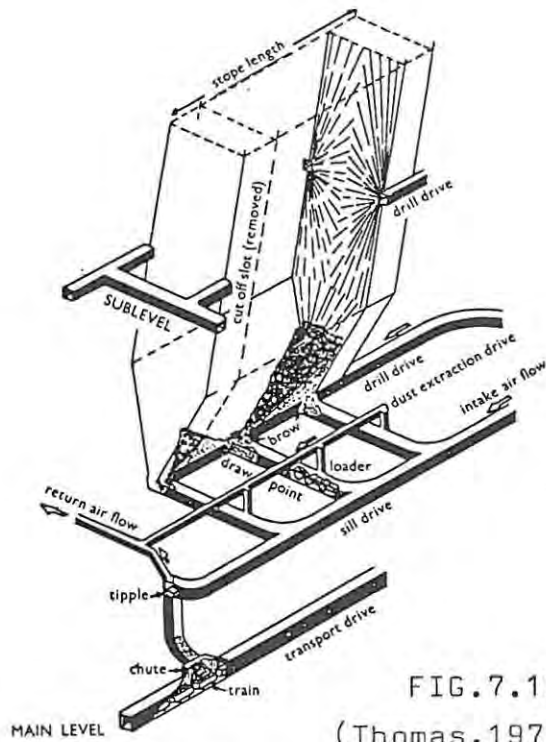Typical section through Victoria 1st level.

FIG.7.10
(Thomas,1978)

Longhole open stoping in narrow and medium width orebodies
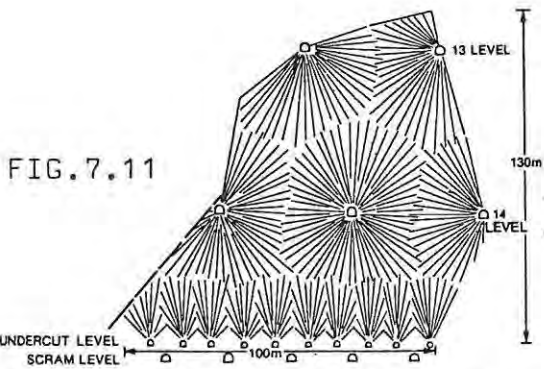(*original courtesy of Mt Isa Mines*)

FIG.7.11

Diagrammatic section of sublevel open stoping, 1100 ore body above 15 level.
(Hornsby & Sullivan,1982)
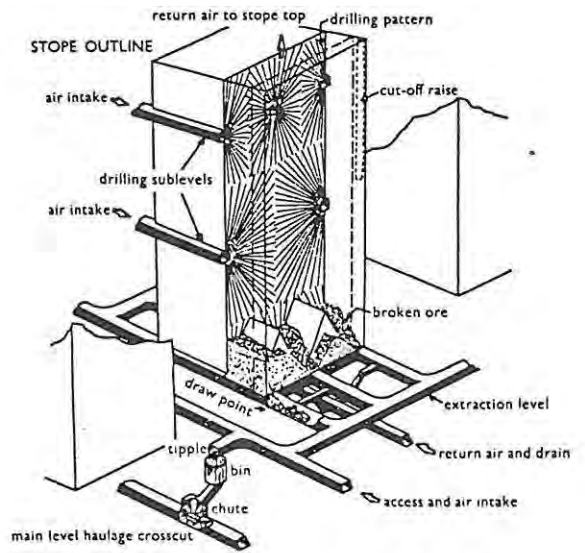
Longhole open stoping in a massive orebody
FIG.7.12   (Thomas,1978)

FIG.7.13

Detail of driller's instruction sheet.

(The Staff, 1957)

A rough understanding of the principles involved in blasting pattern design is necessary to evaluate the mineability of an orebody. Smirnar (1978) described the technique used in iron ore mines at the Urals to estimate the degree of undercharging of the blastholes.   In the

fan of blastholes in Fig. 7.14, the distance a between hole ends is one that results in an optimum concentration of explosives per ton of rock to be blasted (0.15 to 0.20 kg per ton). Near the collar,
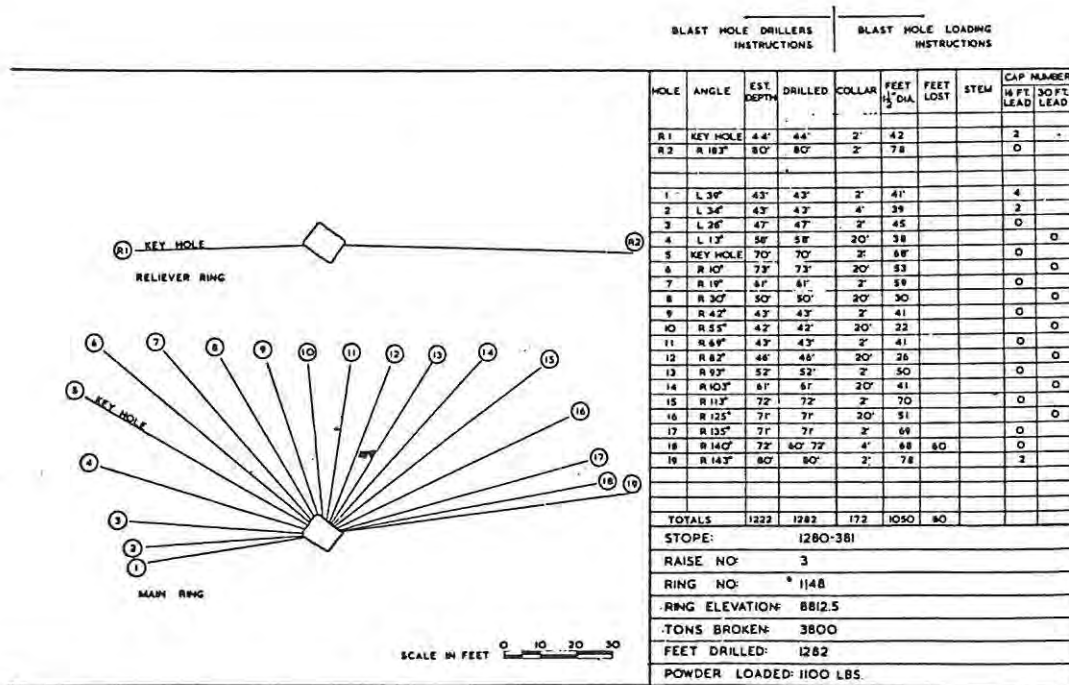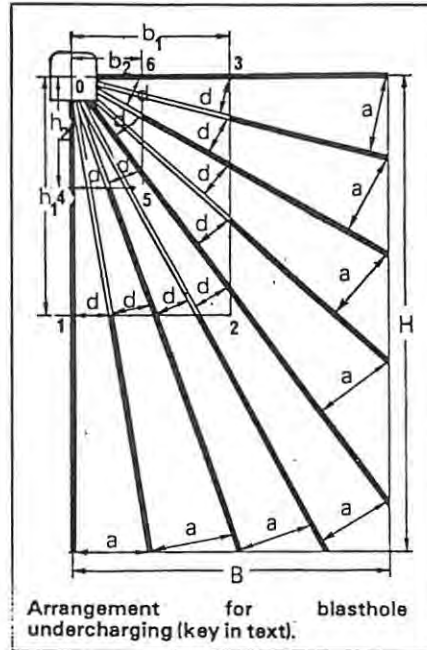

Arrangement for blasthole undercharging (key in text).

FIG.7.14 (Smirnar,1978)

the blastholes must be undercharged to maintain that optimum ratio of explosives to tons of rock. The basic dimension to be calculated is the minimum distance d between charges in the fan. Since the rock layer thickness is constant,$2d$ must be equal or less than a. The line 1-2-3 joins the places where the blastholes are at a distance d from each other, and the line 4-5-6, the places where the blastholes are at a distance $d/2$. Every 4th blasthole within the area limited by the line 4-5-6 is charged, whereas every 2nd blasthole is charged within the area limited by the lines 4-5-6 and 1-2-3. Values of d in the mines of the Urals lie between 1.7 m and 2.2 m, depending on the hardness of the rock.

Transverse sublevel stoping layouts, instead of sublevel caving, have been used in wide steeply dipping orebodies when the hangingwall does not cave readily and when a lower dilution is desired (see Fig. 7.15).

A typical example of the use of transverse sublevel stoping layouts is the 1100 orebody at Mount Isa mines; the orebody was mined in two stages (see Figs. 7.16, 7.17 and 7.18): a primary set of stopes was mined using sublevel stoping, and the pillars recovered using mass pillar blasts into the open stopes, where good hangingwall conditions made it possible, or cut and fill.

FIG.7.15
(Morrison &
Russell,
1973)
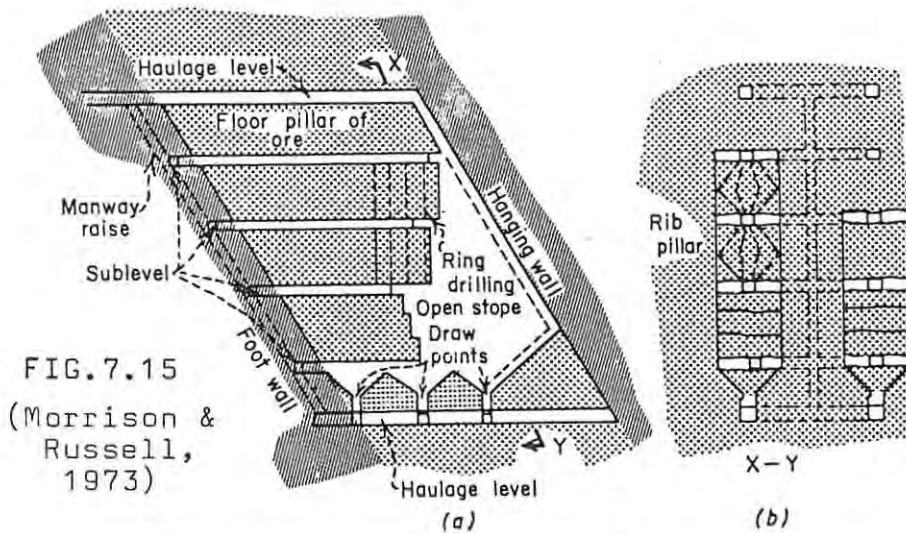
-Sublevel stoping with transverse stopes in wide veins (Jackson and Gardner[7]).
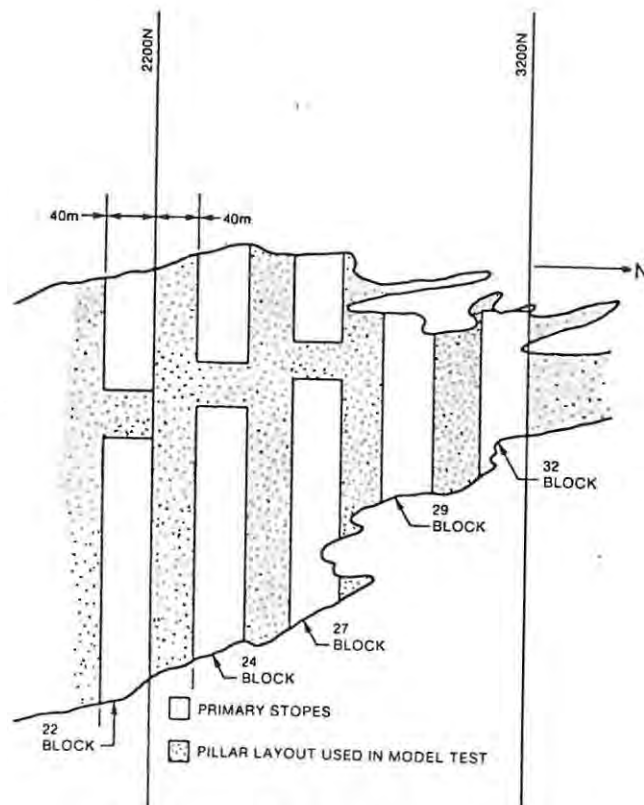
Plan view of the 1100 ore body showing initial layout of stopes and pillars.

FIG.7.16 (Hornsby & Sullivan,1982)
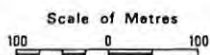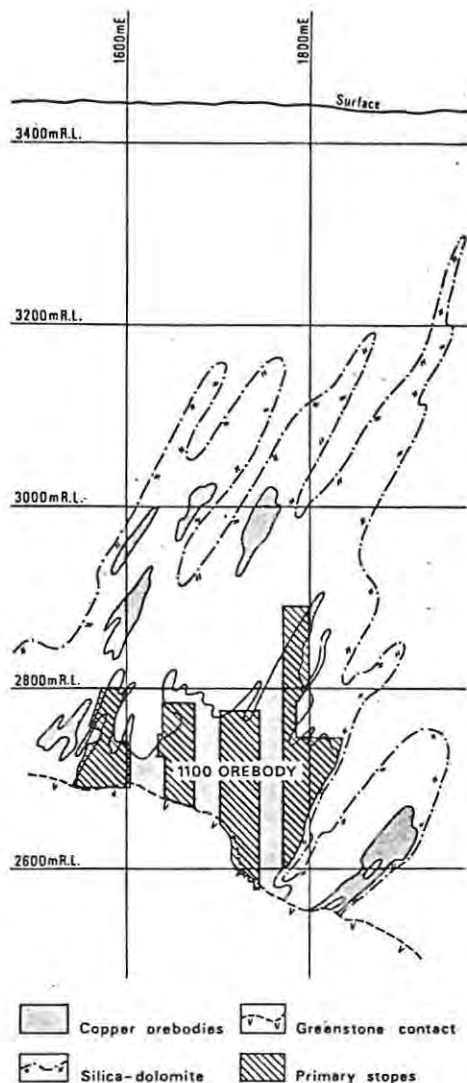
FIG.7.17

> Cross-section through 1100 Orebody showing primary stope and pillar layout

(Kirby et al.,1979)

Idealized two-stage drawpoint extraction in 1100 orebody, Mount Isa Mines
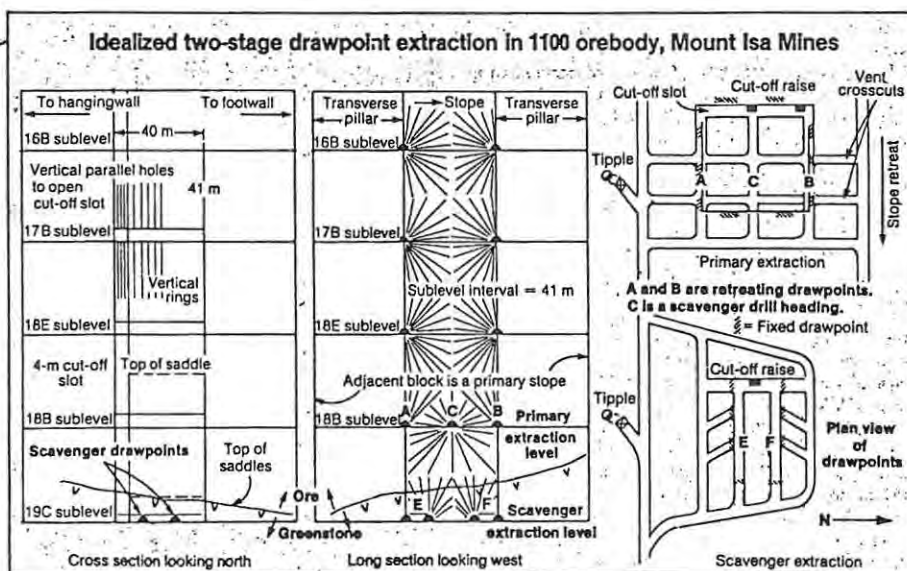
Two-stage draw of ore in 1100 orebody positions primary retreating drawpoints in competent ore on 18B sublevel, leaving crown pillar at C in plan. C rings and E, F rings are then shot to scavenge ore in weak waste.

FIG.7.18

(Dayton,1978)

The "cascade" system of sublevel stoping has been used in the Copperbelt to mine wide, 45° to 25° dipping copper and copper-cobalt orebodies.   Fig. 7.19 shows the method used at Mufulira, while Fig. 7.20 shows the one used at Baluba (a general geological cross-section through the Baluba copper-cobalt deposit is shown in   Fig. 7.23).   At Mufulira (see Fig. 7.19) the ore is broken from a series of 4.2 m x 4.2 m mining drives that follow the footwall ore contact.   The ore passes through the cones into the drawpoint cross-cuts, where it is loaded with LHD units and dumped through ore passes to the main haulage.   The pillars are blasted out as mining retreats.   The method involves waste development (drawpoint cross-cuts) and increasing footwall waste dilution and ore losses as the orebody dip flattens (see Fig. 7.19).
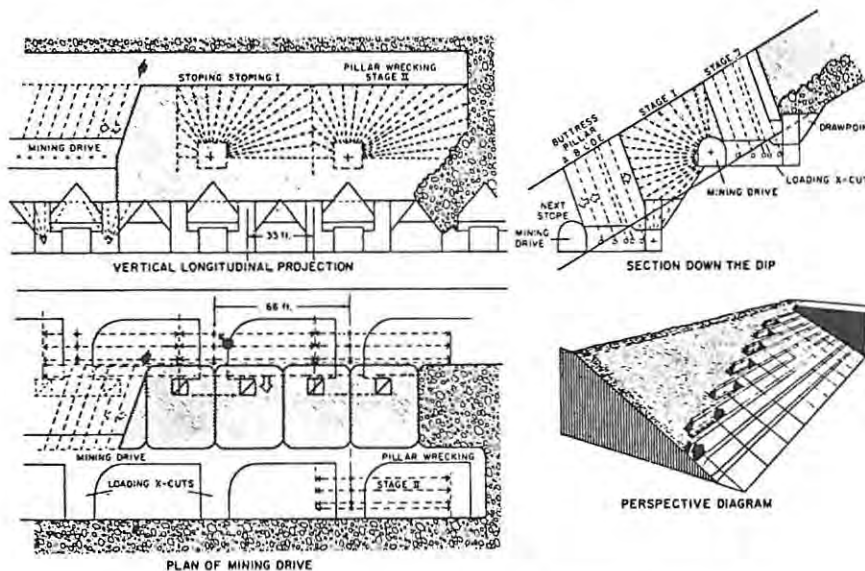
Cascade system of sublevel stoping for dips between 45 and 25°.

FIG.7.19 (Haycocks,1973)
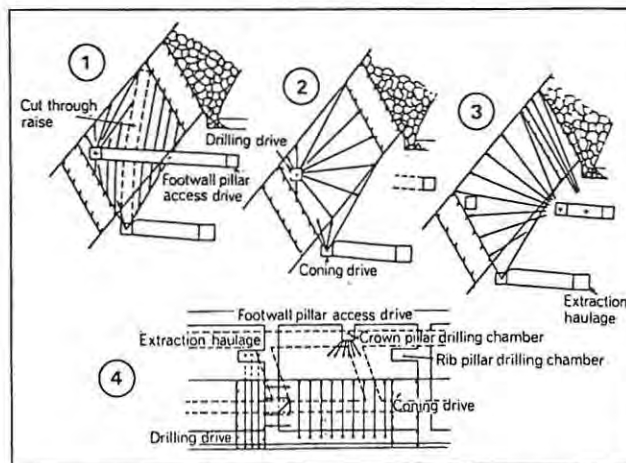
Typical stoping method. 1. Cut through drilling section. 2. Stope drilling section. 3. Pillar drilling section. 4. Composite plan.

FIG.7.20 (Mabson,1976)
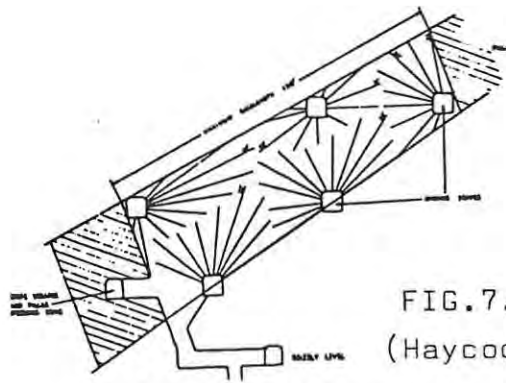
FIG.7.21
(Haycocks,1973)

Plan and section of typical sublevel stopes using a scraper for cleaning.

The method shown in Fig. 7.21, which uses scrapers for cleaning the stopes, was used at Mufulira as well. With this method production is slow due to the time required to set and reset the scrapers.

7.3. Evaluation and grade control

Sublevel stoping is applied to steeply dipping orebodies and diamond drilling during the exploration stage is normally done in cross-section, as shown in Fig. 7.22. The figure shows the way the orebody is delineated during the preliminary stage.

Deeper and more complex orebodies will require additional exploration drilling from underground developments, as it can be seen in Fig. 7.33 which shows a geological cross-section through the Baluba copper-cobalt mine in the Zambian Copperbelt.

The scale of the diamond drilling grid shown in the Figs. 7.22 and 7.23 is seldom enough to plan the final stoping layout and estimate recoverable ore reserves, unless the orebodies are extremely regular in shape and in distribution of the ore within them. The final design of the stoping layout, and the estimation of recoverable ore reserves, generally requires systematic diamond drilling from the footwall haulage drifts, in the way shown in Fig. 7.24. At the Snow Lake gold mine in Archean greenstones of west central Manitoba (Canada), the orebody is outlined on the basis of diamond drill information in vertical cross-
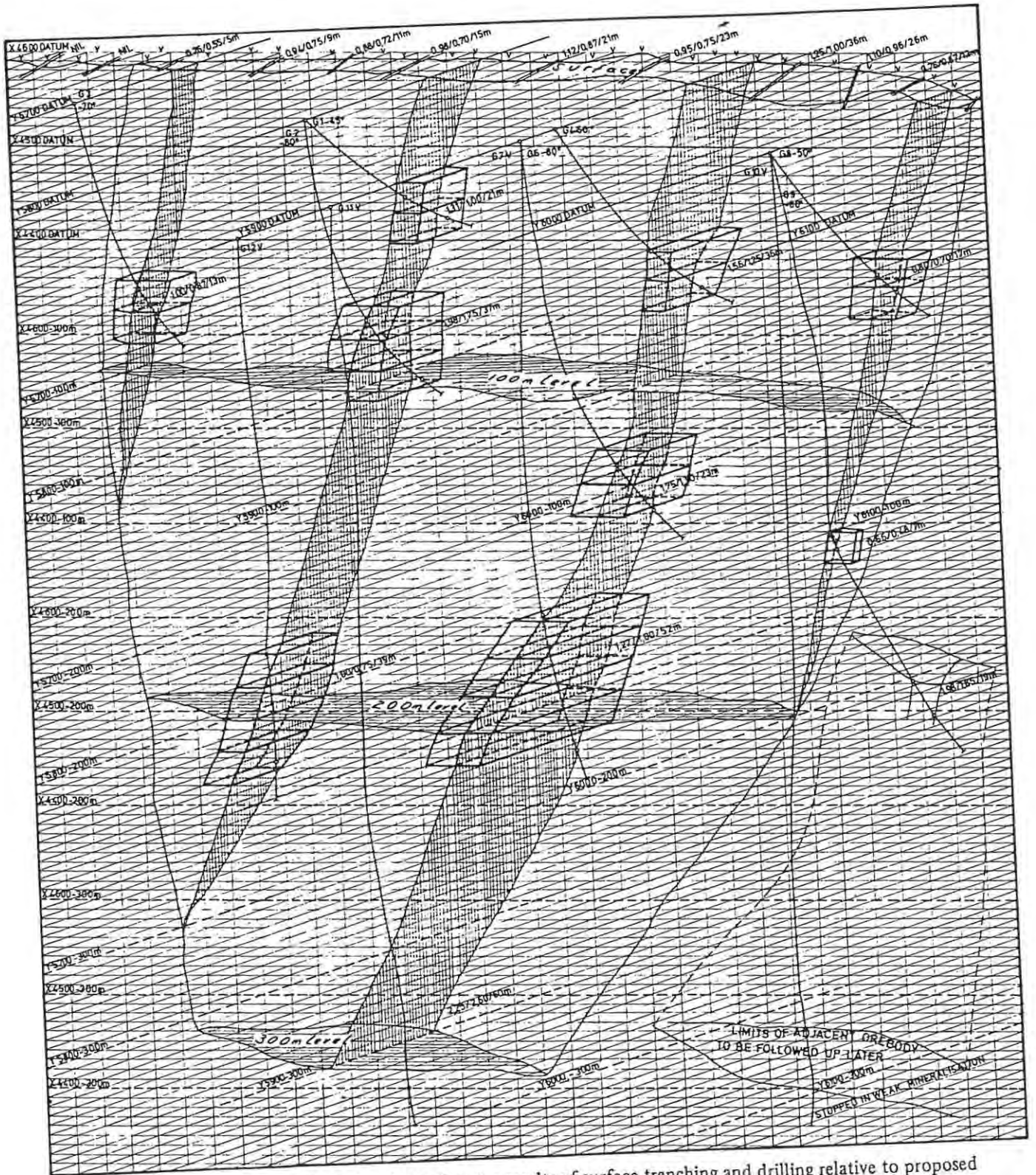
Isometric view of hypothetical orebody showing results of surface trenching and drilling relative to proposed cell grid

FIG.7.22 (Böhmke,1979)

FIG.7.23 (Mabson,1976)
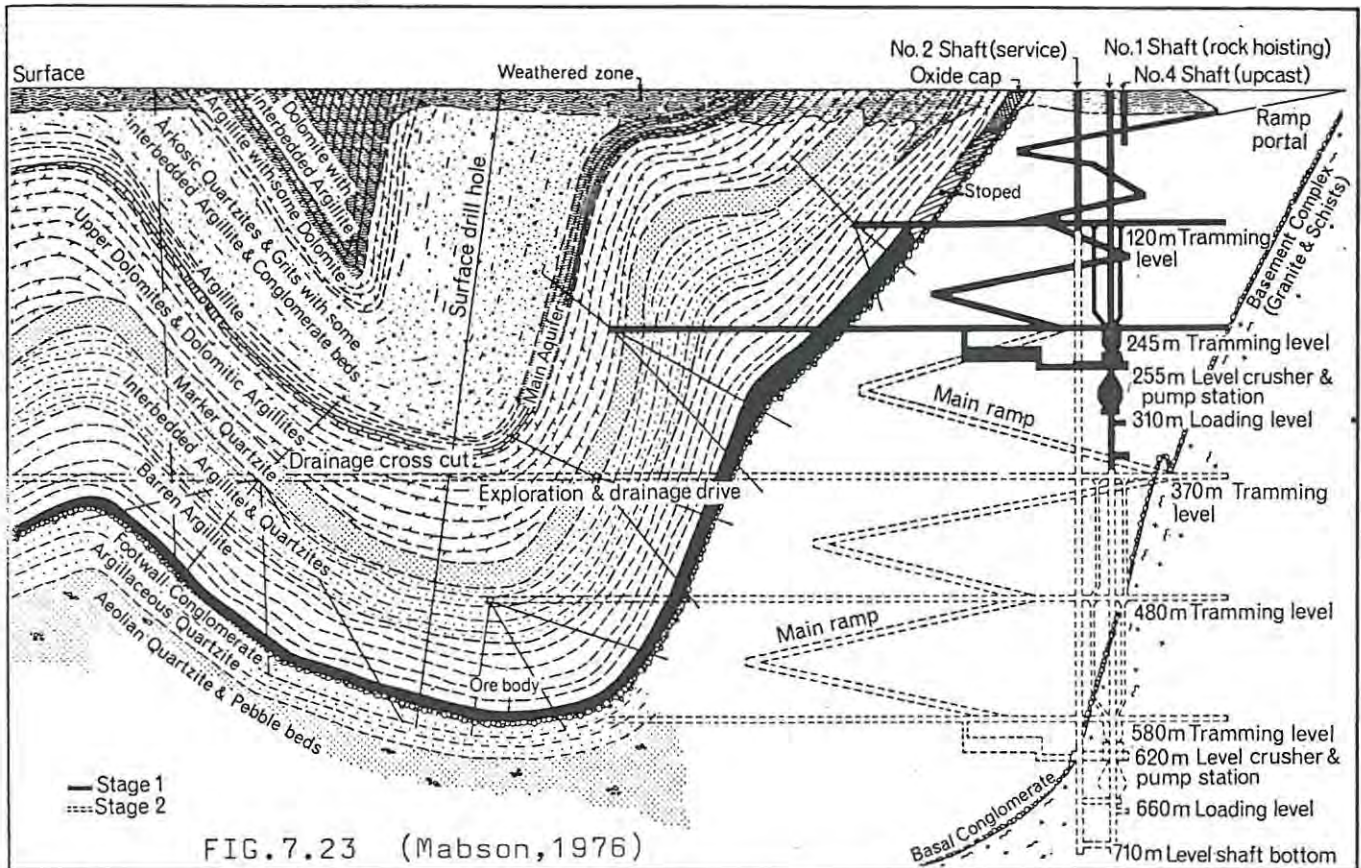
Geological cross section of Baluba mine showing the shaft system and ramp.

sections at 50 ft. ($\pm$ 15 m) intervals (see Fig. 7.24); spacing between cross-sections and vertical distance between borehole intersections in each section, are primarily controlled by the continuity of the mineralization along both directions, as derived from the respective semi-variogram functions. Spacing between these vertical cross-sections normally ranges from 10 to 30 m. At all stages of exploration it is essential to survey the drill holes to obtain an accurate delineation of the orebodies.

Once the orebody has been delineated, the mining layout can be designed based on the criteria mentioned in the previous section. In particular, the downwards gravity flow of the blasted ore must be achieved, and the stope sidewalls must be compatible with the results of a viable blasting pattern. The planned sublevel drifts must be included in the cross-sections in order to visualize the viability of blasting to
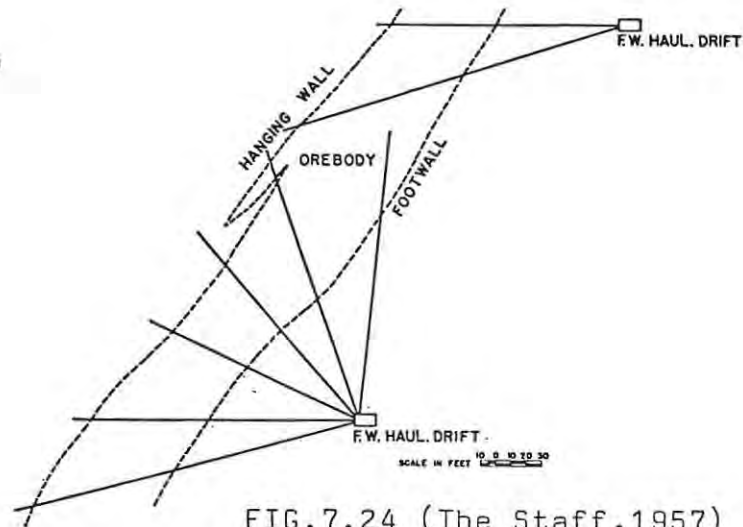
FIG.7.24 (The Staff,1957)

Typical diamond drill section to determine stope limits.

a determined stope sidewall.   Fig. 7.25 shows the planning of a stope
outline on a cross-section showing the outline of the orebody; the
figure does not show the location of the drifts on the different sub-
levels, and the reasons to prefer a particular stope outline in some
areas are not evident.   Fig. 7.26 shows a better example of design
of a stope layout as an overlay to a geological cross-section: breaks
in the slope of the sidewalls take place at the sublevel drifts, or at
half-distance between them (where the fans of blastholes drilled from
both drifts come together, and a break in the stope outline can be
achieved).   Nevertheless, breaks in the slope of the stope outline
like A and B in Fig. 7.26 are possible only if they do not mean to drill
blastholes, from one of the adjacent drifts, to a length beyond the
capacity of the drilling equipment (or to a length at which the hole
deviation makes blasting inaccurate).
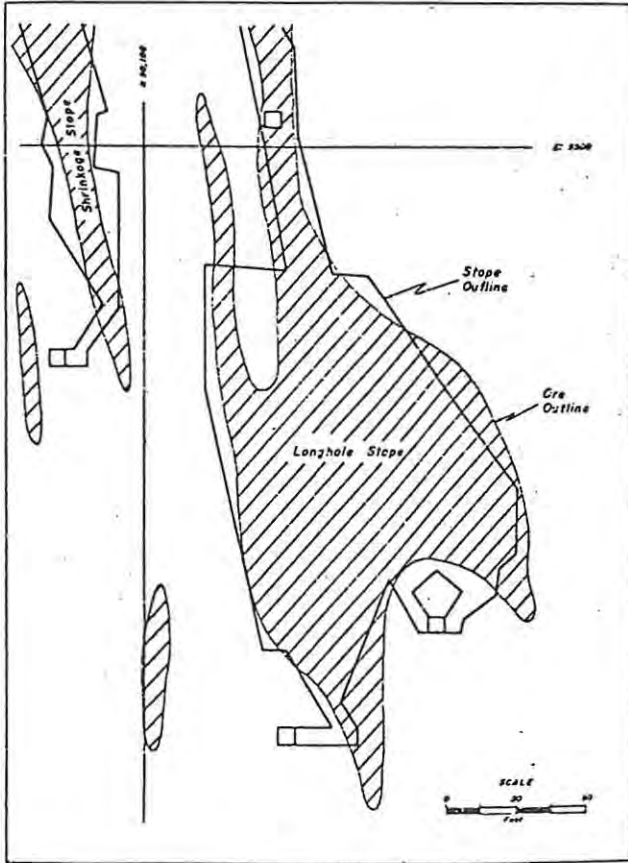
Part of Section 50650E, looking west, at North
Coldstream Mines Limited.

FIG.7.25 (Heim,1968)
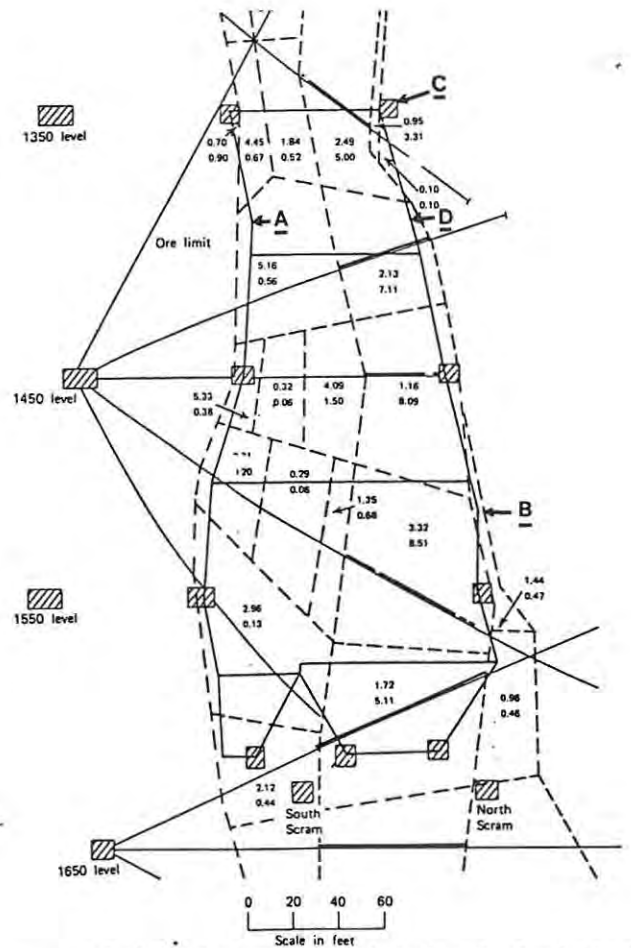
Vertical section through workings in the Geco mine, Ontario, showing
areas of influence assigned to diamond drill holes for tonnage and grade calculations.
(After L. S. Brooks and R. C. E. Bray, "Applications of geostatistics in ore
evaluation," CIM Special Volume No. 9, *Ore Reserve Estimation and Grade Control*,
Fig. 4, p. 181, 1968. Used by permission of the Canadian Institute of Mining and
Metallurgy.)

FIG.7.26 (Peters,1978)

In the drift C in Fig. 7.26 there is no room to drill a blasthole
to produce the stope limit D, which should move further to the right,
thus increasing the dilution. When considering the room in the sub-
level drifts necessary to drill a particular blasthole, it must be
taken into consideration that short blastholes can be drilled with
down to 1.2 m long extension rods (needing less room to work), but
long ($\pm$ 3 m) extension rods are needed to drill long blastholes,
otherwise the hole will deflect and blasting will be inaccurate.

The stope layout must also be optimized in a longitudinal
direction, according to the geology, ore outline and mining criteria.
For this purpose, longitudinal geological sections are drawn (see
example from the Coronation volcanogenic massive copper sulphide
deposit in Canada, shown in Fig. 7.27).

Pockets of waste within the orebody can be left behind as pillars,
same as E shown in Fig. 7.28. This involves slotting again behind the
pillar of waste, and this additional cost must be taken into consideration
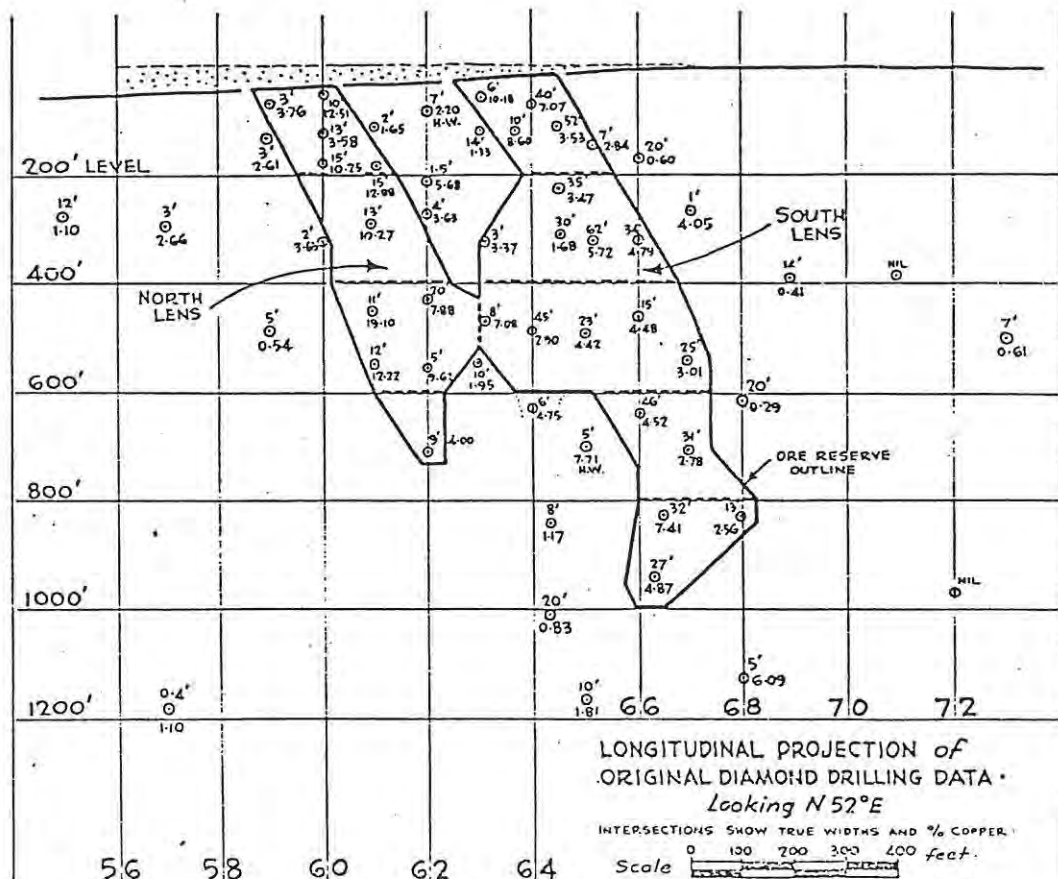
FIG.7.27 (Cairns, 1968)
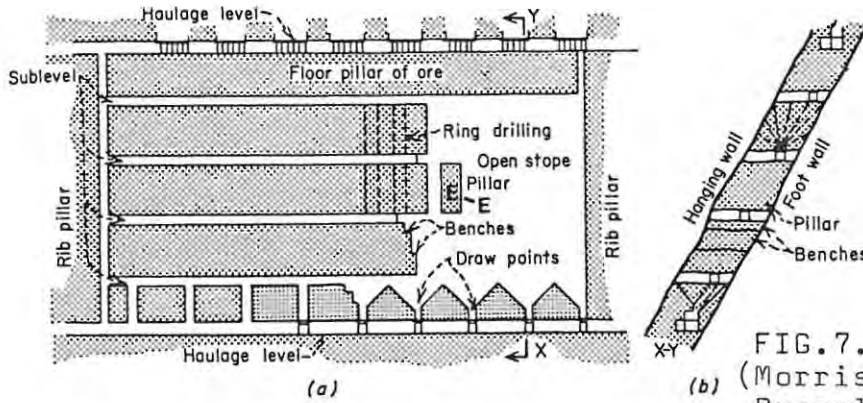
FIG.7.28
(Morrison &
Russell,1973)

Sublevel stoping with longitudinal stopes in narrow veins (Jackson and Gardner[7]).

(there will be a minimum pillar width that justifies the re-slotting).
The pillar can be left between two or more sublevels, and can be shaped
to square, rectangular or triangular shape in a longitudinal section.
Triangular shapes are preferred, because some broken ore is lost on
top of the flat upper surface of rectangular or square pillars.

The sublevel stoping mining method is not flexible to mine
selectively small, narrow veins branching away from the main orebody.
Large, steeply dipping branches of the orebodies can be mined splitting
the stope in two and leaving a pillar between these two stopes.

The vertical cross-sectional method is the most widely used to
estimate ore reserves in sublevel stoping. Fig. 7.26 shows the way
borehole intersections are given areas of influence in a cross-section,
according to the distribution of the different ore types, and criteria
such as limiting the influence of a sample to the half-way to the next
sample on the same ore type. When the mining layout is superimposed,
recoverable ore reserves can be estimated.

To make easier the comparison of the ore reserve estimates with
the actual production, the estimates of recoverable ore reserves are
made stope by stope. Partial estimates for different mining units
(developments, crown, bottom and rib pillars, undercutting, stoping)
within each block can be obtained from sets of sections similar to the
one in Fig. 7.26; each mining unit has a different mining recovery,
dilution, cost and time-position in a mining schedule, and the partial
estimates make production planning easier.

Fig. 7.29 shows the pattern of blasthole rings in a longitudinal section through a stope, and a slice (production unit) according to the standard blast pattern. An alternative pattern is shown in Fig. 7.30. The ore reserves to be recovered during the stoping stage can be further subdivided in "reserves per slice", using cross-sections and longitudinal sections with the blasting sequence. Thus predictions of the daily grade and tonnage produced per stope can be made, and production scheduling and ore blending becomes easier.

The volume of the ore between two consecutive sections is estimated using the formula for a frustom of a cone, unless the layout on a longitudinal section or on a plan indicates something different. The formula used is:

$$V = \frac{h}{3} \left(S_1 + S_2 + \sqrt{S_1 S_2} \right)$$

where S_1 and S_2 are the areas of the orebody on the respective adjacent sections, and h is the distance between these sections. (The formula for the volume of a cone, $V = Sh/3$, is used for the extremities of a zone).
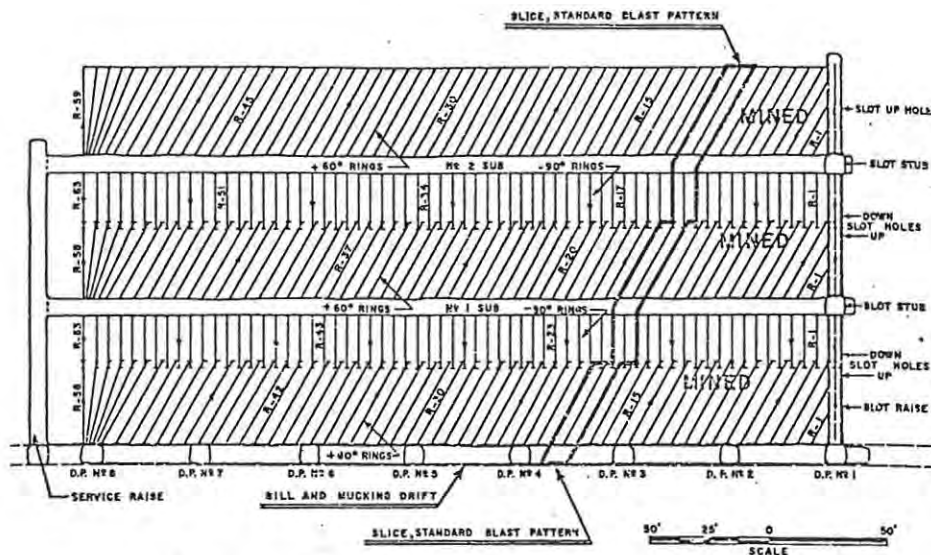
Idealized Longitudinal Section of a Standard Longhole Stope. Mining starts at the bottom of the block, and the free face at each horizon is kept slightly behind the next one above.

FIG.7.29 (Rancourt & Evans,1968)
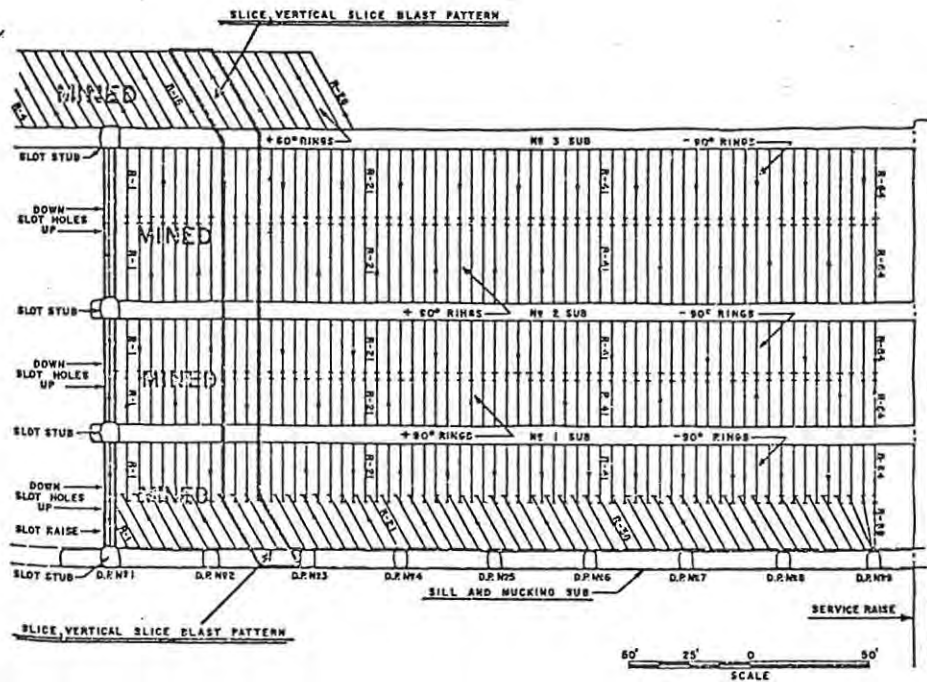
Idealized Longitudinal Section of a Stope where Vertical Slices are taken from Top to Bottom.

FIG. 7.30 (Rancourt & Evans, 1968)

Grade (see example from Mount Isa in Fig. 7.31) and grade x thickness contours on a cross- and/or longitudinal sections, and polygonal (see example from the Anglo-Rouyn copper-gold deposit in northern Saskatchewan, in Fig. 7.32) methods have been used to estimate recoverable ore reserves in sublevel stoping operations. They do not give good results: the polygonal method does not take into consideration the geology, and the contouring method assumes lineal continuity of the grade between adjacent samples, what is not always true.

The main grade control problems in sublevel stoping operations are the overbreaking of waste and underbreaking of ore. To a lesser extent, the grade control is affected by the caving of the hangingwall and slabbing of the sidewalls, and by losses of broken ore during drawing. A 15% to 20% overall dilution is frequent, unless good grade control is applied.
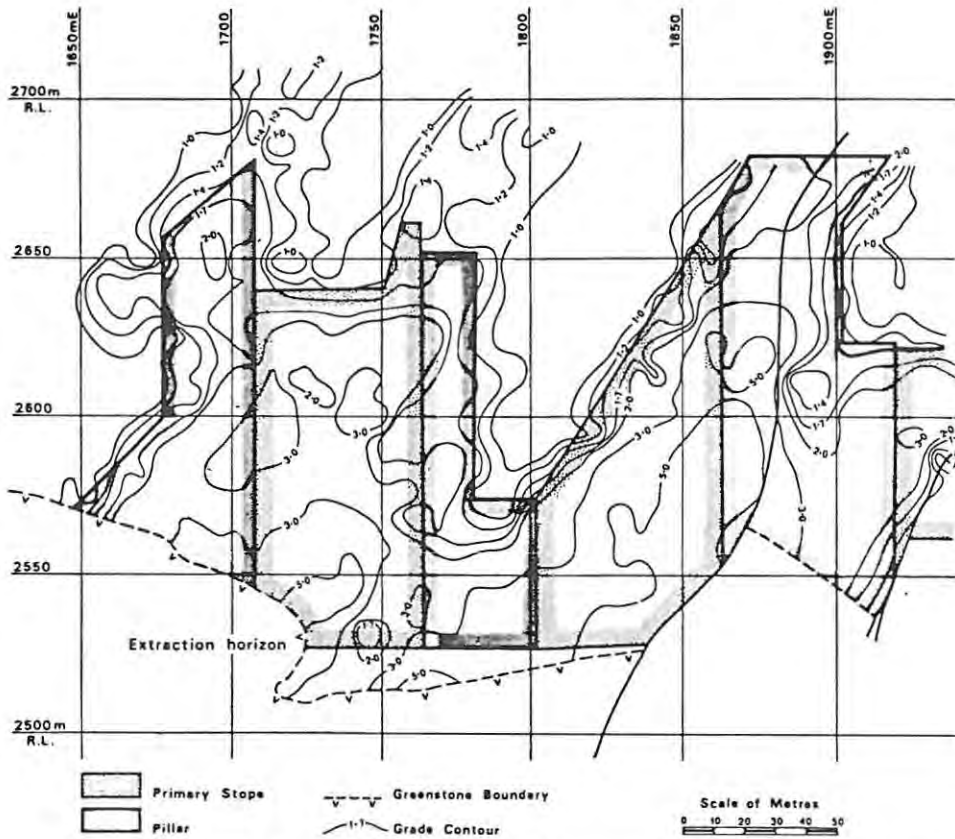
Cross-section showing computer-contoured assay values and primary stope/pillar design (assay values deleted for clarity)

FIG.7.31 (Kirby,Blair & Hutton,1979)
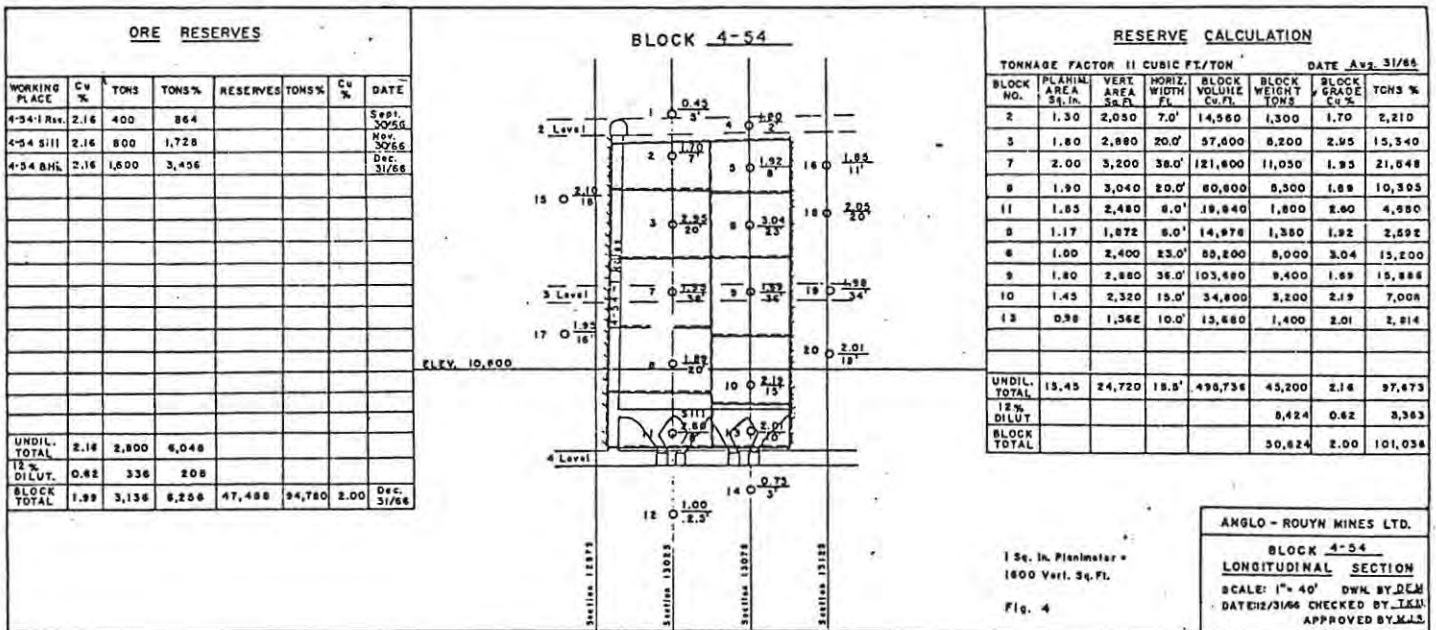
Example of an Ore Reserve Tabulation for a Stope Block.

FIG.7.32 (Skopos & Lawton,1968)

Some dilution is introduced during the development of the drifts
on the sublevels. The drifts following the waste-ore contacts should
ideally keep that contact some 30 cm from the sidewall of the drift.
Those 30 cm of waste should be all the dilution taken. But with
irregular orebodies it is difficult to keep that relationship.
To avoid the risk of developing the drifts entirely in ore, at distances
of about 1 m from the contact, which would result in underbreaking of
ore and lower ore recovery during the stoping stage, the irregular
waste-ore contacts are frequently kept in the middle of the drifts.
The ore mined in the development of sublevel drifts can be as much as
25% of the total, and this form of dilution can be significant.
Surveying the diamond drill holes to produce accurate geological plans
and mining layouts, and routine supervision of the developments by the
mine geologists help to reduce this form of dilution to more acceptable
levels (no more than 5 to 10%).

To reduce the overbreaking of waste, the blastholes delineating
the sides of the stope should be planned parallel to the stope sidewalls,
and not perpendicular or in an angle to them. These blastholes parallel
to the sidewalls of the stope reduce the damage that blasting produces in
them, thus decreasing the amount of dilution by caving and slabbing of the
sidewalls. The control of the overbreaking in the stope on the right in
Fig. 7.33 is going to be worse than the one in the stope on the left,
because the former is going to be cut with blastholes in a steep angle to
its sidewalls.
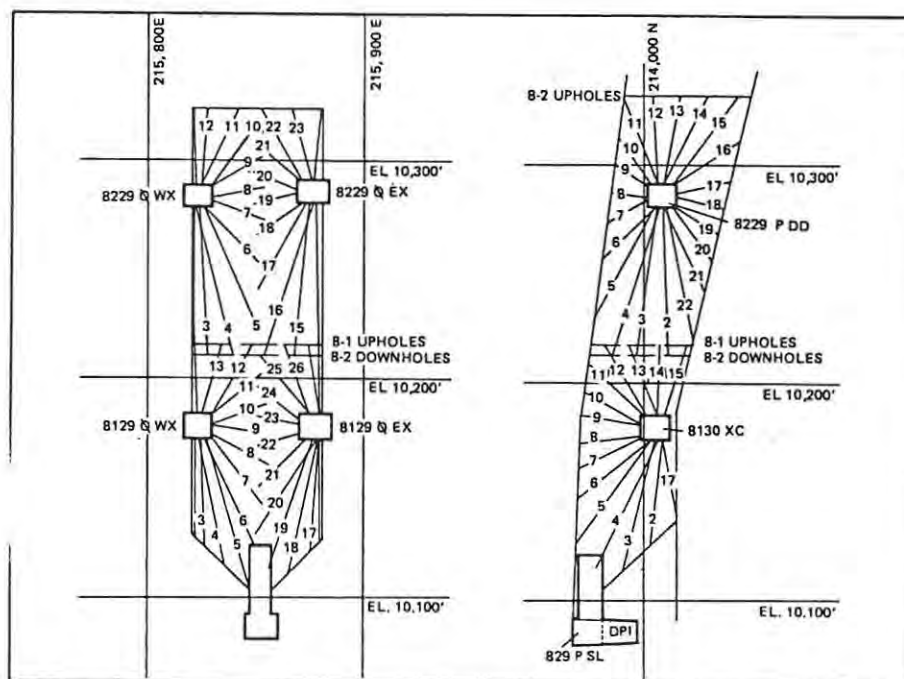
FIG. 7.33   (Blakey, 1979)

The recovery of broken ore is maximized by (i) optimum drawpoint spacing, (ii) steep slopes of the sidewalls of the stope ensuring downwards gravity flow of the broken material, and (iii) optimum blasthole pattern design and blasthole loading practice, to produce the correct fragmentation and to avoid flyrock and the spread of broken ore to already abandoned cones (which could be full of caved waste).

If we know where in a stope the production came from, and if we have ore reserve estimates for that part of the stope (say, for example, slices 8 to 12 of the undercutting), mine factors (to be applied to the rest of the estimates of that, or similar, mining unit, which has characteristic dilution and ore recovery) can be derived by "reconciliation" of the estimates with the production. The estimates can be "reconciliated" with grab samples of the muck or with the mill headgrade. In the second case, we know the tonnages sent to the mill from different stopes and the estimated grades of these stopes, and an overall estimated grade can be calculated. This overall estimated grade is compared with the mill headgrade and the difference corrected, proportional to the estimated grade and the produced tonnage of each stope, using a "grade factor" as shown in the following example:

Stope	Estimated Grade (G)	Tonnage (T)	G x T	Corrected Grade (G x G.F.)
A	2%	2000 tons	4000	1.68%
B	3	1000	3000	2.52
C	1.5	3000	4500	1.26
D	1	1000	1000	0.84
Overall	1.78%	7000 tons	12500	1.5%

Mill head   = 7000 tons @ 1.5%

G x T   = 7000 x 1.5 = 10 500

Difference = 10 500 - 12 500 = -2000

Grade Factor (G.F.) = 1 - 2000/12 500 = 0.84

The previous example assumes that the dilutions and ore losses of the different stopes are similar. If we want a more accurate grade factor, only materials of similar dilution and ore recovery characteristics should be included in a "reconciliation". Otherwise a "double reconciliation", mill headgrade/muck samples and muck samples/estimated grades, seems to be a more correct approach.

This type of reconciliation is normally applied to the estimates of recoverable ore reserves, in order to estimate the dilution and ore losses affecting them during mining, but it can be used to estimate some specific forms of dilution as well. For example, the dilution by mining waste during development can be measured and applied to the original estimates, and the "reconciliation" is going to account for the other forms of dilution only.

As "mill headgrade" we can use the average grade of the automatic sampling of the mill discharge, but the best is the one that has been corrected by means of a metallurgical balance:

Sampling mill discharge = 1.6%
Tonnage milled (dry) = 7000 tons
Concentrates (dry) = 360 tons @ 2.5%
Tailings (dry) = 6640 tons @ 0.25%
Calculated headgrade = (360 x 25 + 6640 x 0.25)/7000 = 1.52%

7.4. Costs and productivities of sublevel stoping

In a review of the copper mining industry, R.C. Howard-Goldsmith (Mining Magazine, Feb. 1978, pp.111) quotes an average cost of $4.00 (in 1978 american dollars) for sublevel stoping operations. This average cost can vary within a wide range of values, because of the number of alternative mining layouts that can be used, and the large variety of orebodies and orebody-shapes that can be mined with sublevel stoping.

Laurence (1982) gives the following break-down of mining costs for an average trackless sublevel stoping mine:

Development 30%
Load and haul 20%
Supervision and service 14%
Stoping 11%
General 6%
Hoisting 5%
Power 3%
Stope fill 1%

To give an idea of the range of costs and productivities, Pugh and Rasmussen (1982) analyzed two different steeply dipping veins - a 10 ft wide one and an 18 ft wide one - and two alternative developments for each one.

The 3 m (10 ft) wide vein was supossed to be developed in the way shown in Fig. 7.34: 45.7 m long stopes, with four 12 m apart sublevels, a 4.5 m spacing between drawpoints, and a grizzly level plus scraper. An alternative using a 1 cu.yd. diesel LHD instead of grizzly level and 25 hp scraper has been considered.

The 5.5 m (18 ft) wide vein is assumed to be developed in the way shown in Fig. 7.35: 40 m long sublevels at 33,5 m vertical spacing, and footwall developments designed for the use of trackless rubber-tired mucking equipment (LHD units). Two alternatives were considered: the use of 2 cu. yd. and 5 cu. yd diesel LHDs.
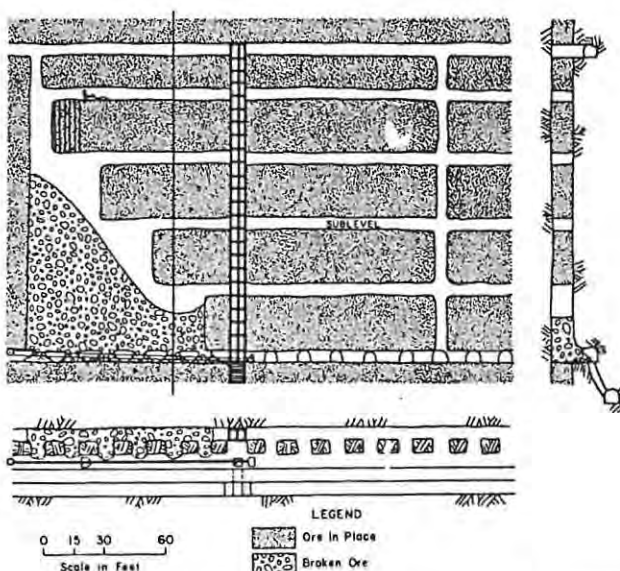

LEGEND
Ore in Place
Broken Ore

0 15 30 60
Scale in Feet

FIG.7.34 (Pugh & Rasmussen,1982)

LEGEND
Ore in Place
Broken Ore
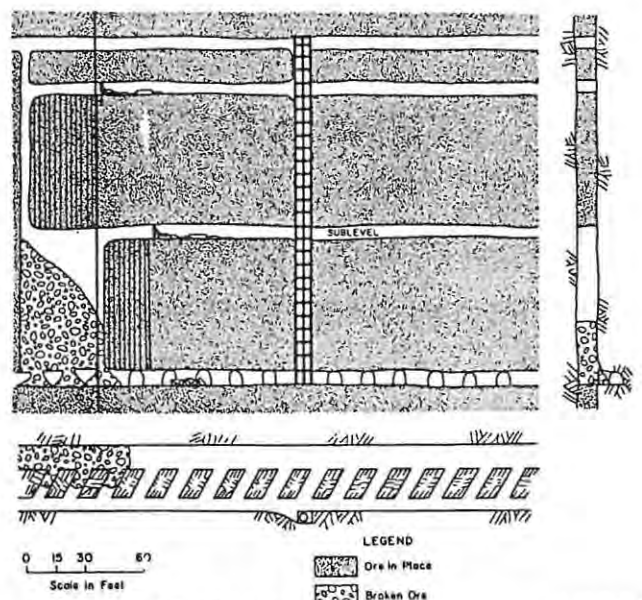
0 15 30 60
Scale in Feet

FIG.7.35 (Pugh & Rasmussen,1982)

The detail of productivities and costs (in 1976 american dollars) for the different alternatives is summarized in the Table in Fig. 7.36.

Sublevel Long-Hole Stoping—Costs and Productivities, 1976 Dollars					
		Vein Width 10 ft		Vein Width 18 ft		
Unit Operation	Unit Cost, $/st of Ore	Small Diam Long-Hole Stopes With Scram Drift Developed Above Haulage, Good Ground Conditions		Large Diam Down-the-Hole Long-Hole Stope With Mechanized Loading, Good Ground Conditions		Qualifications
		A	B	C	D	
Development						8-hr shift, 6.5 hr effective time at face.
Case I 10-ft* vein	4.08	4.08	4.08	—	—	
Case II 18-ft vein	1.49	—	—	1.49	1.49	Labor rate $10.50 per hr includes bonus and fringe.
Drill and blast						
2-in.* diam downholes, 40 ft long, 10-ft vein 388.8 st*, 2-in. diam	1.29	1.29	1.29	—	—	Unit operation costs include labor.
4-in. diam down-the-hole, 100 ft long, 18-ft vein 2332.8 st, 4-in. diam	0.97	—	—	0.97	0.97	Equipment costs include maintenance parts and labor, tires, and fuel. Equipment depreciation is not included.
Mucking						All materials are based on current 1976 prices.
Slusher—25 hp*	1.10	1.10	—	—	—	
LHD 1 cu yd*—diesel	0.92	—	0.92	—	—	Machine ownerships costs excluded.
LHD 2 cu yd —diesel	0.70	—	—	0.70	—	
LHD 5 cu yd —diesel	0.64	—	—	—	0.64	
Total Cost Per St		$6.47	$6.29	$3.16	$3.10	
Stope Crew Size		3.0	3.0	3.0	3.0	
Productivity St Per Miner shift		38.1	49.2	82.1	99.2	

*Metric equivalents: 1 in. × 25.4 = mm; 1 ft × 0.3048 = m; 1 cu yd × 0.764 554 9 = m³; 1 st × 0.907 184 7 = t; 1 hp × 745.699 9 = W

FIG. 7.36 (Pugh & Rasmussen, 1982)

7.5. Vertical crater retreat

The vertical crater retreat method (VCR) is a variety of sublevel stoping that has been gaining popularity in recent years, both as a stoping method and for pillar extraction. The method is based on the concept of cratering by the blasting of a spherical charge.

A crater blast is a blast when a spherical charge is detonated beneath a surface, which extends laterally in all directions beyond the point where the surrounding material will be affected by the blast.

Livingstone (1973; in: Lang, 1982) determined the relation between the energy of the explosive and the volume of material affected by the crater blast, which can be expressed by the empirical equation:

$$D_b = \Delta E W^{1/3}$$

where d_b is the distance from the surface to the center of gravity of the charge (i.e. the depth of burial); Δ is the ratio d_b/N, in which N is the critical distance at which breakage of the surface above the spherical charge does not exceed a specified limit; E is the strain energy factor, a constant for one given explosive-rock combination; and W is the weight of the explosive charge. When d_b is such that the maximum volume of rock is broken to an excellent fragment size, this burial is called optimum distance (d_o).

Through cratering tests, the empirical relationship between the depth ratios ($\Delta = d_b/N$) and energy levels (crater volume to weight of explosive charge ratios) shown in Fig. 7.37 can be obtained. From these empirical curves, production-scale blast can be designed to satisfy any demand (from simply isolating the rock from its surroundings to the effective breaking of the rock). Fig. 7.38 shows an empirical curve obtained by cratering tests in massive sulphides of the Centennial copper-zinc mine, near Flin Flon (Croker, 1979).

Not all the ores will crater successfully. Cratering characteristics are unique for each ore and crater design must be based on test work.

When spherical charges are detonated in a downwards direction, towards the back or hangingwall of a mine opening, gravity instead of having adverse effects enlarges the crater dimensions by removing the entire rupture zone.
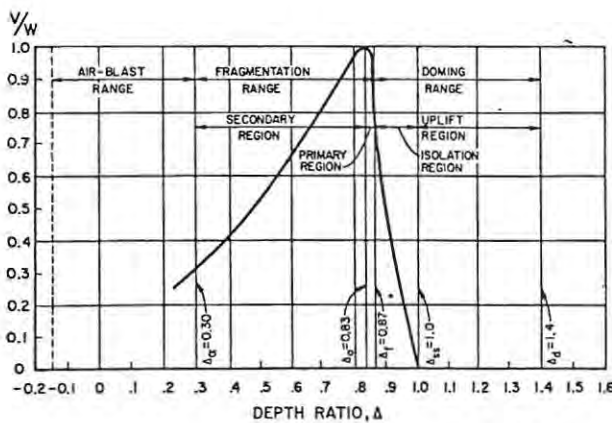
Cratering test in which depth ratios are plotted against energy levels. This idealized curve of Livingston also shows the important transition limits from the isolation range to the other extreme which is the air-blast range.
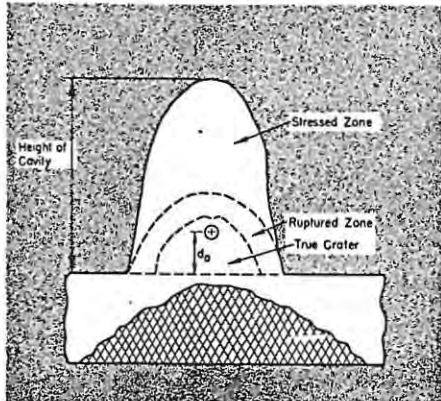
FIG.7.37 (Lang,1982)

FIG.7.38 (Crocker,1979)

Crater tests. Four in. holes of 4 to 10 ft length were drilled in drift wall, each loaded with 10 lb Hydromex M210U and primed with 0.7 lb Procor, and blasted
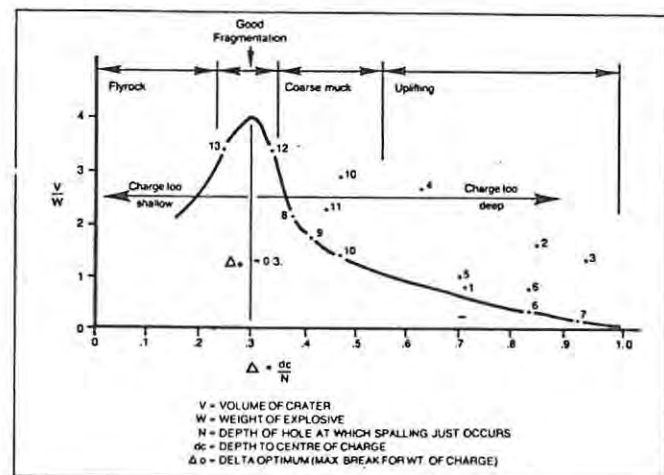
Depending on the stability of the rocks, the blast can damage the rocks within the tensionally stressed zone that normally occurs above an underground opening, and produce caving; the caving may enlarge several times the cavity that would result from cratering (see Fig. 7.39). Conveniently orientated geological discontinuities in the hangingwall will considerably increase the effectiveness of crater blasting.

Depending on rock properties and structural geology, the total height of the cavity may exceed the optimum distance of the spherical charge from the back many times.

FIG.7.39   (Lang,1982)

Comparison of Spherical and Cylindrical Charges

	Spherical charge*	Cylindrical charge*
Type of explosive	C2	C2
Charge weight	10 lb	10 lb
Hole diameter	4½ in.	2-5/8 in.
Diameter to length ratio	1:2.7	1:15
Volume of crater	155 cu ft	38.6 cu ft
Crater radius	5.7 ft	4.8 ft

* Metric equivalents: 1 in. x 25.4 = mm; 1 ft x 0.3048 = m; 1 cu ft x 0.028 316 85 = m³; 1 lb x 0.453 592 4 = kg.

FIG.7.40   (Lang,1982)

The shape of the charge has a great importance in the breakage process, as it can be seen in the experimental comparison between blasting spherical and cylindrical charges shown in Fig. 7.40. Cylindrical charges do not produce a very effective fragmentation, and it was only with the introduction of large diameter (165 mm, i.e. 6 1/2 in) blastholes to underground mining operations that it has been possible to develop a mining method - vertical crater retreat - from the principles of cratering. As long as the deviation from the true spherical charge (diameter = length) is not greater than 1:6 diameter to length of charge ratio, the results are practically the same as those of a true spherical charge.

In vertical crater retreat large diameter (6 1/2") holes are drilled on a designed pattern from drifts above a stope to bottom in the back of the undercut. Spherical charges are placed within these holes at the calculated optimum distance from the back of the undercut. The charges

are detonated and a horizontal slice of hangingwall is blasted
downwards into the undercut.   The procedure is repeated and
mining of successive slices retreats in a vertical upwards direction.

The method (i) eliminates raise boring, slot cutting and dilution
of the ore by backfill; (ii) greatly improves fragmentation; (iii)
reduces labor and time requirements; (iv) eliminates uphole drilling
and blasting; and (v) minimizes or completely eliminates the damages
by blasts to the walls and retreating backs of the stope.   A slot is
needed only for the undercut, which is mined by conventional methods.

The stope layout shown in Figs. 7.41 and 7.42 is being used at the
Kidd Creek mine (Blakey, 1979) between the 370 m and 500 m levels.
The drawpoint design is based on the capabilities of the ST-8 Wagner
scooptrams.
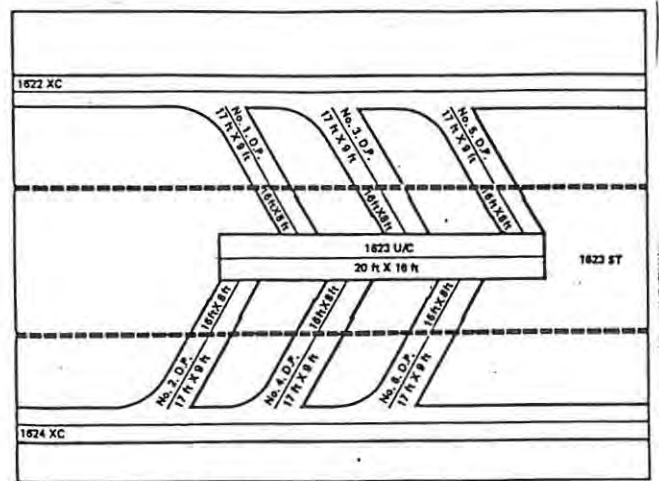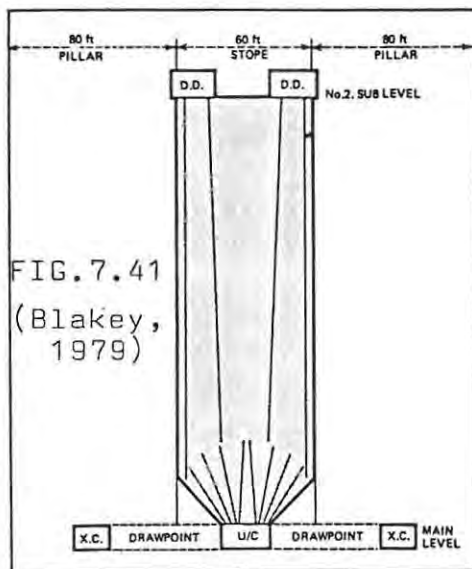
FIG.7.41 (Blakey, 1979)

FIG.7.42 (Blakey,1979)

The selectivity of this method is limited.   Pockets of waste within
the orebody cannot be left behind as pillars and must be blasted and drawn
as dilution to the ore.   The possibilities of adjusting the stope outline
to the irregularities in the shape of the orebody are more limited than
when using sublevel stoping, but the method still offers some flexibility
as it can be seen in Figs. 7.43 and 7.44.

Figs. 7.43 and 7.44 show the typical vertical crater retreat
mining layout used to mine the copper skarn deposit at the Whitehorse
mine, Yukon (Janssens and Percival, 1981). On September 30, 1980,
with extraction at 78% of the reserve tonnage, the copper recovery was
66% of the reserves, representing a dilution of 15.4% (assuming a 0% Cu
grade for the waste).
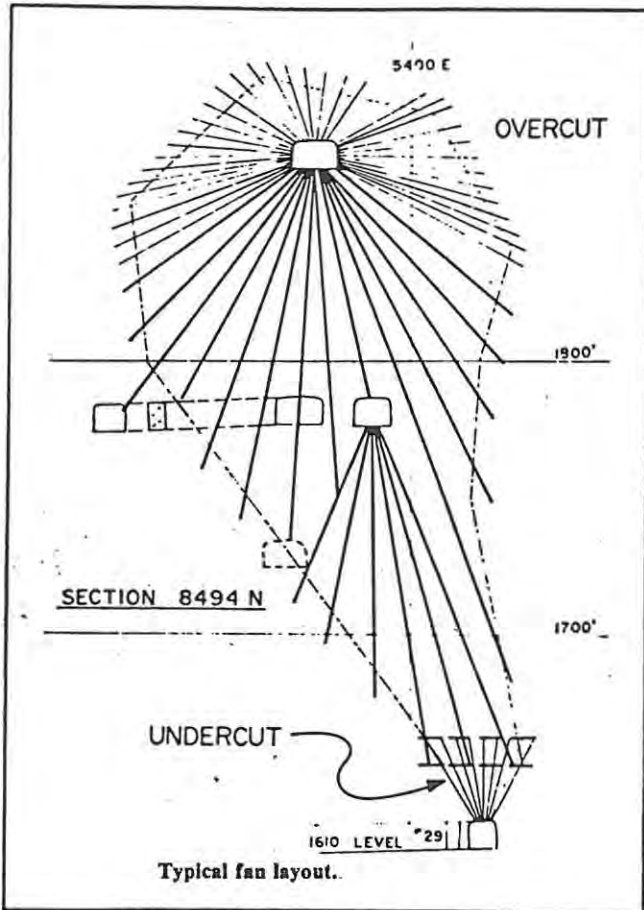
FIG.7.43 (Janssens & Percival,1981)
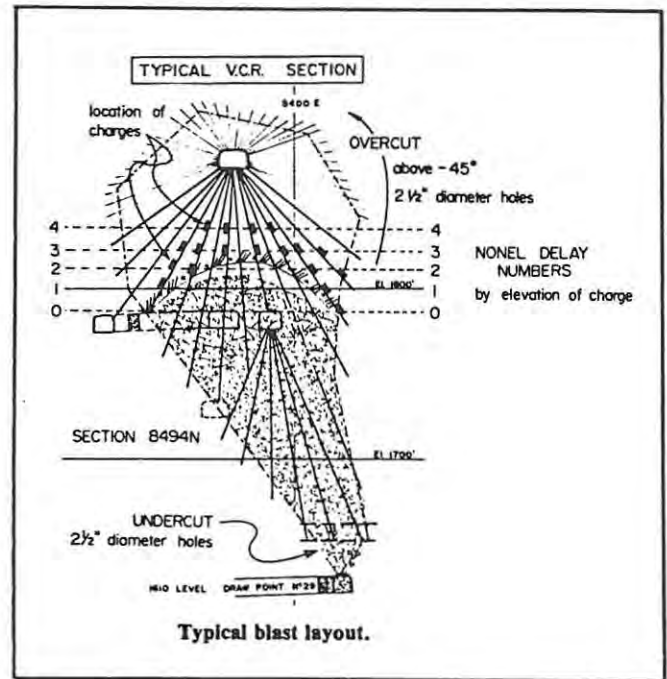
FIG.7.44 (Janssens & Percival,1981)

The method has been successfully used to mine narrow (1.5 to 6.0 m
wide) steeply dipping massive sulphides at the Radiore No. 2 copper-zinc
mine, Quebec (Goodier, 1982). Fig. 7.45 shows a typical sequence of
crater blasting in a cross-section through a stope, and the table in
Fig. 7.46 summarizes the drilling and blasting costs (in 1982 Canadian
dollars). The powder factor is a low 0.63 kg of explosives per ton
blasted.

Vertical crater retreat can be used to mine the ore left at the bottom of an open pit, in the way shown in Fig. 7.47, making use of the surface drilling equipment. At Ardlethan, NSW (Australia), the Carpathia orebody, a small offspring of the main tin orebody, was found about 40 m below the surface just outside the open pit mine. The body was developed and mined by vertical crater retreat, drilling the blastholes from the surface (see Figs. 7.48, 7.49 and 7.50).

FIG.7.45 (Goodier,1982)
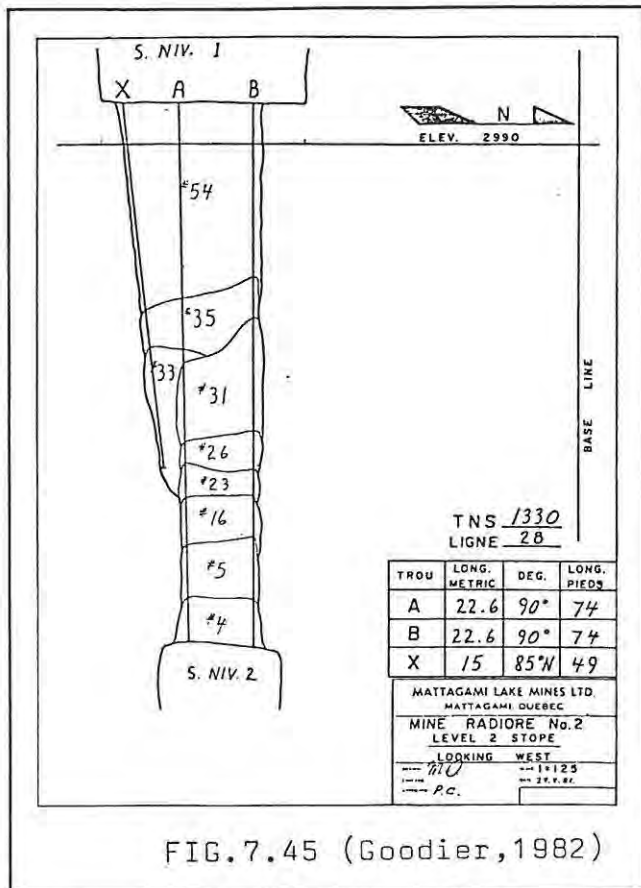
Drilling and blasting costs

Cost per Foot Drilled:

Drilling Labour	$ 4.18
Drilling Material	0.80
Equipment Rentals	3.47
Maintenance and Fuel	2.18
Blasting Labour	4.28
Explosives	5.93
Blasting Material	1.46
TOTAL COST:	$22.30
TONS/FOOT:	7.19
COST/TON:	$ 3.10

FIG.7.46 (Goodier,1982)

A crown pillar of ore left at the bottom of an open-pit mine is extracted by the VCR method.

FIG.7.47
(Lang,1982)
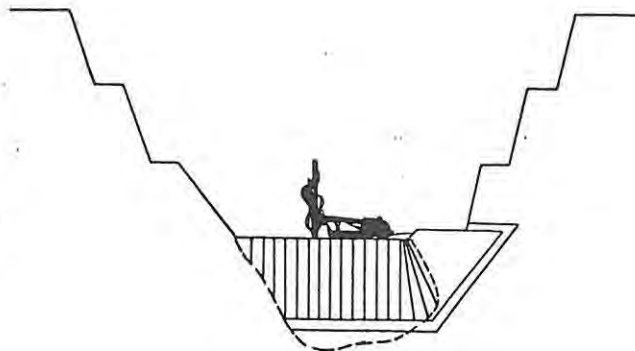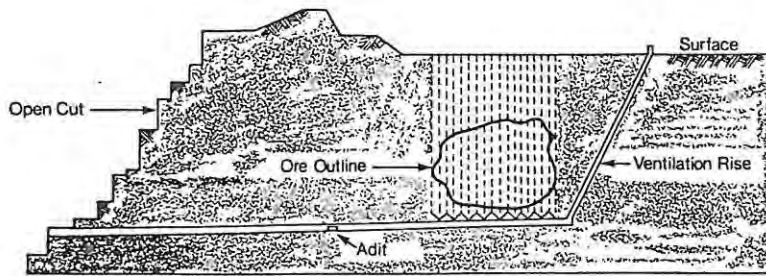
The Carpathia ore body is drilled from the surface for the VCR method.

FIG.7.48
(Lang,1982)
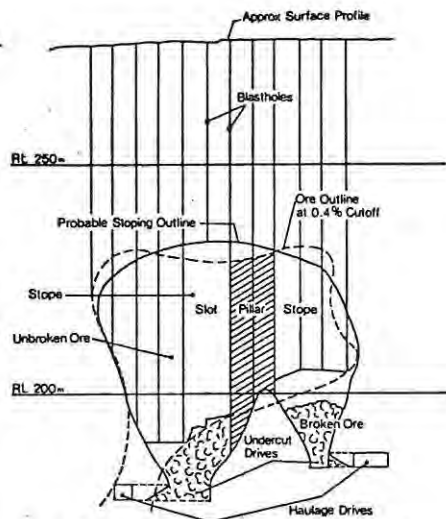
Cross section of the Carpathia ore body shows the development work and the sequence of the VCR mining.

FIG.7.49 (Lang,1982)
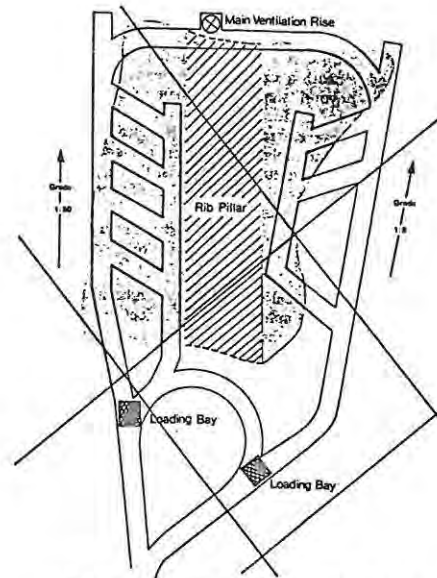
Plan view of the VCR operation at the Carpathia operation.

FIG.7.50 (Lang,1982)

7.6. Strassa

At the Strassa iron ore mine in Sweden (Mining Magazine, December, 1974; and Björkstedt, 1982) a mining system was developed during the 1960's to mine wide, steeply dipping orebodies, with stable hanging and footwall and competent ore. The hangingwall would not cave easily and the orebodies are not very suitable for sublevel caving mining. The method is inexpensive, efficient and results in lower dilutions than sublevel caving.

The orebodies are mined using trackless sublevel stoping, in a transverse layout consisting of a series of 50 m high x 22m wide stopes, separated by 12 m wide pillars and leaving a 10 m thick crown pillar

above the stope.   Drilling is done from the undercut and from a
sublevel 25 m above the footwall.   Once the sublevel stoping stage
has finished, the rib (above the first sublevel) and crown pillars
are blasted from two sublevels : 25 m and 52 m above the footwall.
The bottom portion of the rib pillars (below the first sublevel) is
recovered by sublevel caving (at this stage the hangingwall caves).
The overall dilution is about 20%.   The layout is summarized in
Fig. 7.51.

A similar method has been used at the Madelaine mine, Quebec
(Gaumond and Parfitt, 1971).   Chalcopyrite and bornite occur as
stockwork in hornfels-altered metamorphic aureole to a paleozoic
granitic stock.   The sulphides have cemented the fractured rock,
which stands well and the overbreak is very minor.

The mining method adopted at Madelaine is a modification of the
system developed at the Strassa mine.   The ore is mined in a series
of transverse open stopes, with rapid removal of the pillars, retreating
from the top down.   The layout was designed for trackless equipment
(Wagner ST4 scoop-trams), with levels at 180 ft ($\pm$ 55 m) intervals
connected by a spiral ramp on a 20% grade (see in Figs. 7.52 and 7.53
this typical layout of accesses on a trackless mine).

Procedure for the mining
method for the 195 to 255-m
levels.

FIG.7.51

(Björkstedt,1982)
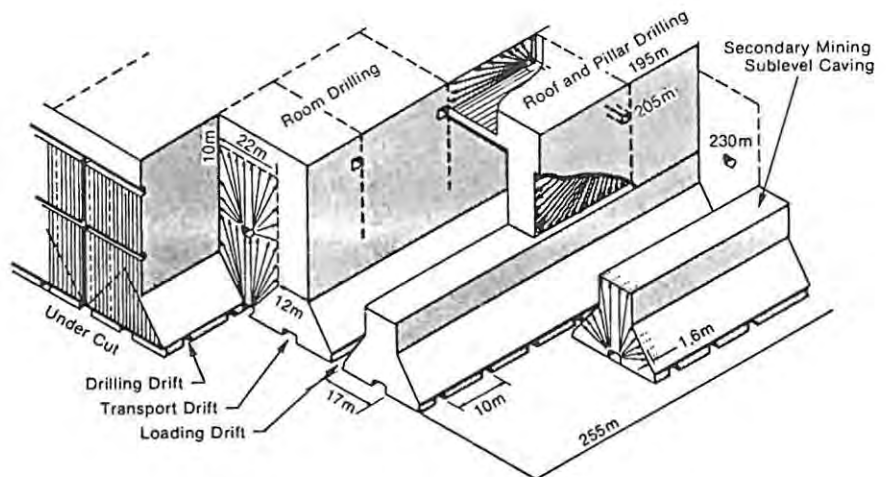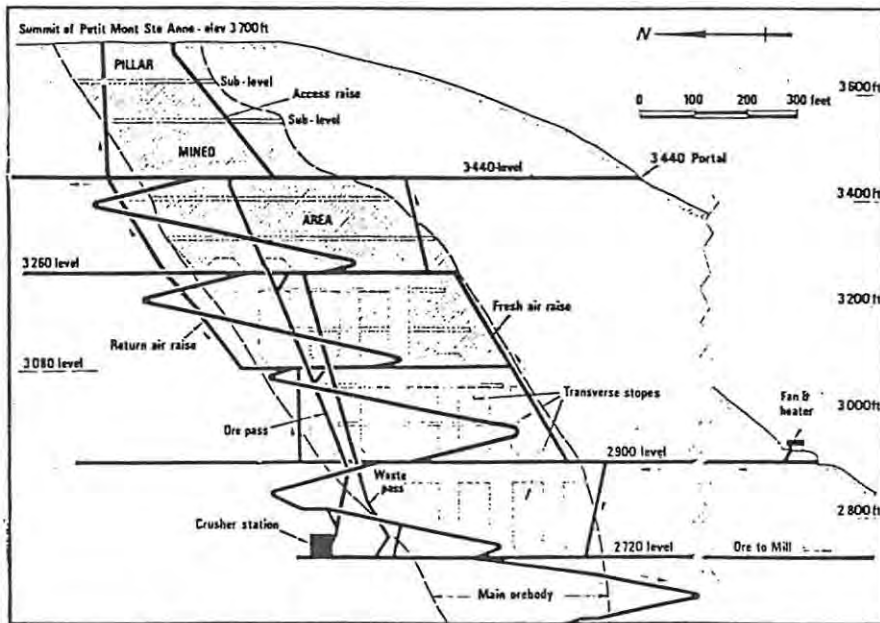
Development of main orebody – longitudinal section looking east

FIG.7.52 (Gaumond & Parfitt,1971)
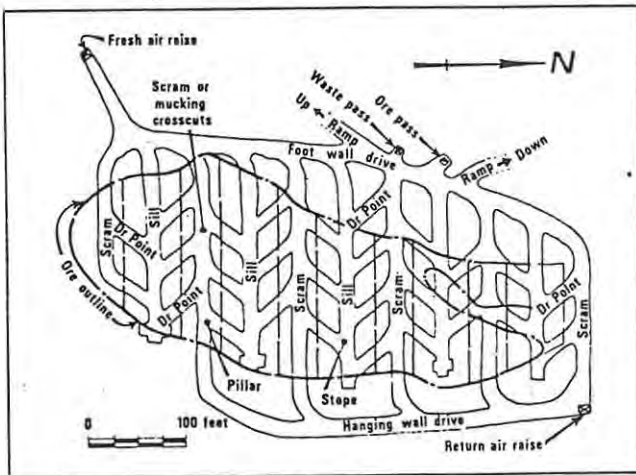
Stope development on the level consists of driving parallel crosscuts through the orebody on 50ft centres

FIG.7.53 (Gaumond & Parfitt,1971)
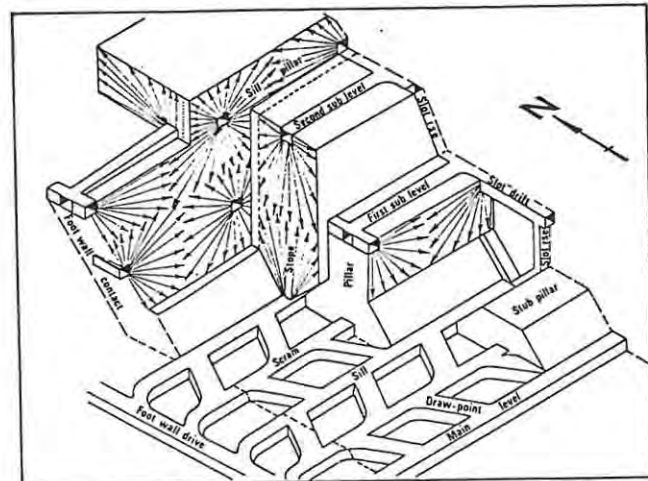
Schematic representation of Madeleine mining method

FIG.7.54 (Gaumond & Parfitt,1971)

Figs. 7.53 and 7.54 show the stoping layout. The orebody is mined by a series of transverse longhole stopes (sublevel stoping), 150 ft (45.7 m) high and 60 ft (12.3 m) wide, spaced on 100 ft (30.5 m) centres. This leaves 40 ft (12.2 m) wide temporary pillars between

stopes and a 30 ft (9.1 m) sill pillar below the level above. Two sub-
levels are established at 75 ft (22.9 m) and 150 ft (45.7 m) above the
footwall; they are developed from the ramp. The stopes are fan-drilled
from the undercutting in the footwall level and from the sublevels,
while the rib and sill pillars are drilled, in a second stage, from both
sublevels in the way shown in Fig. 7.54. The explosive consumption
during the stoping stage is 0.2 kg x ton, normal in sublevel stoping,
and decreases slightly to 0.18 kg x ton during the pillar blasting.

The method gives 100% recovery of the ore. About 60% of the tonnage
is recovered during the stoping, with minimal dilution, and 40% during the
pillar blasting. The productivity is about 16.5 tons per total man shift
and 48 tons per underground man shift. The mine operating costs per ton,
in 1971 american dollars, are as follows:

Mining, engineering and geology .... $ 1.92
Milling .... $ 1.58
Services and administration .... $ 0.90
$ 4.40

8.   ROOM-AND-PILLAR

8.1. The application of room-and-pillar

Room-and-pillar is an open stoping mining method applied to sub-horizontal or flat-dipping orebodies, which consists in opening multiple stopes or rooms, leaving solid material to act as pillars to support the cover load.   Since the dip of the orebodies is not steep enough to have gravity flow of the broken ore to a drawpoint or gathering point, the material must be loaded in the room where it was extracted or moved by mechanical means (mainly scrapers) to a drawpoint.

The application of this method requires tabular subhorizontal orebodies, economically mineable over a stope height (or width of the orebody) of at least 0.8 m to 1.2 m, and with a competent hangingwall that can be economically supported by pillars.

The method has a high degree of flexibility to adapt to changes in the orebodies.   However, until such a change is indicated many aspects of the mining cycle are repetitious, and mechanization becomes easier (labour constitutes only about 40% of the total mining costs) and productivity fairly high.   The system can be applied as a very selective mining system, leaving waste in pillars, or raising and lowering the hanging or footwall of the stope to follow the contours of flat orebodies of changing thicknesses.   Yet the same system can become a bulk mining system, taking everything at a given horizon and thickness and leaving only uniform pillars.

Room-and-pillar can be applied to several levels simultaneously, and, when the ore reserves are large enough, it is usually very easy to develop a large number of stoping areas producing simultaneously (see example of mine development at the Fletcher mine in Fig. 8.1).

The amount of development which will open sufficient working areas can be kept to a minimum prior to production, and since most of this development is in ore some mineral production will result from them.
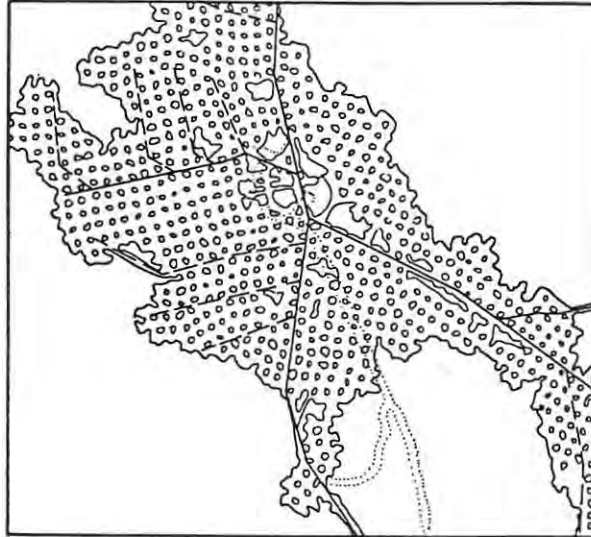
Mine development at Fletcher mine, showing symmetric development to allow maximum number of stopes to be placed into operation in minimum time. Shaft is located as close to center of ore-body mass as possible (adapted from Bullock, 1973).

FIG.8.1 (Bullock,1982)

The main disadvantage of room-and-pillar is the necessity of maintaining a stable roof over active areas for long periods of time; a large roof area is exposed during the development stage. Incompetent hangingwalls can be extremely costly to support. The increase in cover load upon the support pillars with depth makes necessary to increase the size of the pillars, thus resulting in lower returns and ore recoveries. Mechanized, trackless room-and-pillar operations can be "capital intensive", since nearly every job is mechanized requiring a very large capital expenditure outlay for equipment. The higher capital cost is normally compensated for by lower operating cost per ton of ore produced and by lower development cost.

8.2. Pillar supports

Assuming that the overlying rocks behave as an isotropic elastic material, the vertical stress over a pillar will correspond to the cover load, and the horizontal stress will be the corresponding lateral stress that will be generated by Poisson's ratio for the material and degree of constraint that is imposed upon the pillar.

At the edge of a pillar the horizontal stress will be zero, and the yielding of the pillar sides, as a result of the imposed load, will be opposed only by the tensional strength of the pillar material and by the friction generated between the yielding pillar material and the strata in contact with it, above and below. The restraint imposed upon the pillar material increases in intensity from zero at the pillar edge to a maximum somewhere within the pillar, and in a large pillar its centre will be completely restrained and will be able to support any load. Hence, a small number of larger pillars is preferred to a great number of small pillars of equivalent total area. Similarly, a square pillar is stronger than a rectangular pillar of the same area.
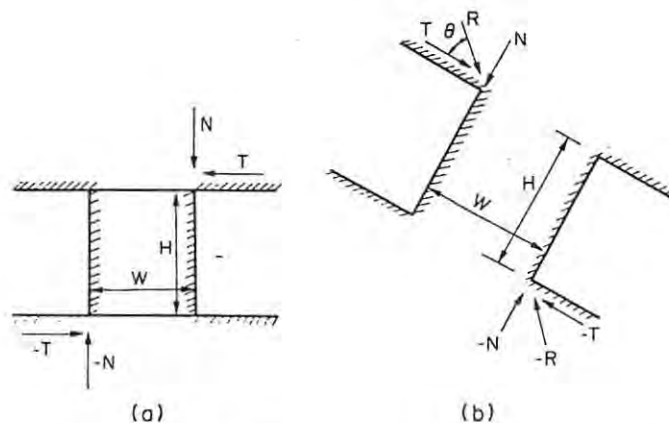
Forces acting upon a pillar in inclined strata: (a) horizontal deposits; (b) inclined seams.

FIG.8.2 (Roberts,1982)

In inclined orebodies (see Fig. 8.2.b) the effect of gravitational forces acting down the dip must be considered. An overturning couple will be created at the top of the raise side of the pillar, and this is opposed by a couple acting at the base of the pillar on the dip side. To counter this effect, the pillars in inclined strata should be rectangular, with the short axis horizontal.

A simplified estimate of the probable load on the pillars, i.e. the average unit vertical stress on a pillar σ_p , can be stated as follows:

$$\sigma_p = \frac{(\text{mined area}) + (\text{pillar area})}{(\text{pillar area})} \times \sigma_v$$

where σ_v is the total vertical stress. This is a square power law, and for extractions above 40% to 50% the load on the remaining pillars rises very rapidly.

A safe load on a pillar is expected to be about 50% of its maximum calculated strength. The compressive strength S of a square pillar in massive elastic rock, with a width to height ratio (W/H) different than one and within the range $0.5 < W/H < 4$, can be estimated using the formula:

$$S = s (0.778 + 0.222 W/H)$$

where s is the measured compressive strength of a representative specimen of rock with a W/H ratio of 1. Fig. 8.3 shows the variation of the compressive strength with the H/W ratios in different rock materials.
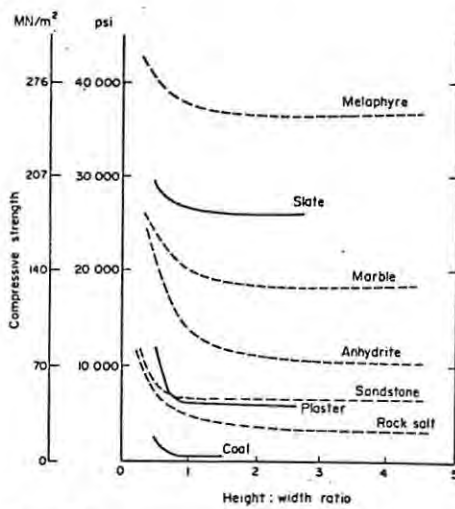
Variation of compressive strength with *H/W* ratio in different rock materials (Eynon [19]).

FIG.8.3 (Roberts,1981)
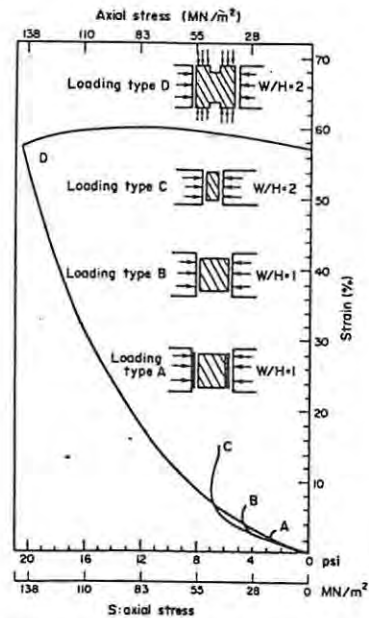
Behaviour of pillars in various loading conditions (Serata [21]).

FIG.8.4 (Roberts,1981)

The conditions of end restraint on pillars under load have a pronounced effect on their stability; a rock pillar situated between a hard-rock roof and floor will be stronger than a similar pillar interposed between, say, weak shales and clay sediments. Fig. 8.4 shows the compressive strength of different model pillars for different

end restraint conditions. Type A restraint simulated low-friction and conditions, while types B and C had a larger lateral friction which doubled the fracture strength of the pillar. In type C the width to height (W/H) ratio of the pillar was doubled and this trebled the fracture strength. Finally, when the ends of the specimen were confined by rings (type D), the fracture strength was increased to eight times the uniaxial value (type A).

Rock discontinuities, such as jointing, bedding planes, faults, unconformities and cleavages induced by mining, reduce the shearing and tensile strength of the rocks, thus affecting the stability of the pillars and the hangingwall of the openings. The orientation of these discontinuities is an important factor. Horizontal discontinuities in subhorizontal openings reduce the friction and the restraints of the pillar ends, in the way shown in Fig. 8.4 (type A). Discontinuities dipping at angles close to 45° (see Fig. 8.5) considerably reduce the shear strength of the pillar. Nearly vertical discontinuities reduce the tensile strength of the pillar, and in a room-and-pillar layout the advance of the rooms should be in a direction perpendicular to that of the cleavage or other discontinuities of this type (see Fig. 8.6).

stability affected by jointing
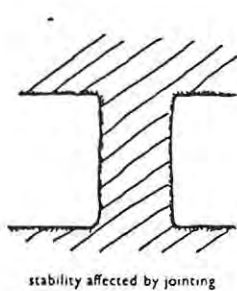
FIG.8.5
(Thomas,1978)
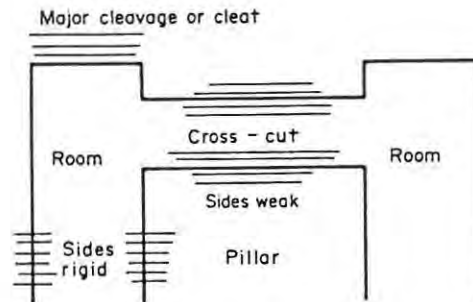
Effect of cleat direction on the relative strength of pillar sides in rooms and cross-cuts.
FIG.8.6 (Roberts,1981)

Discontinuities affect the stability of the hangingwall of the openings as well. Fig. 8.7 shows some examples of roof failure at the White Pine copper-silver mine, Michigan; note the effect the increase in opening width has on a discontinuity that occurs in the hangingwall, close and parallel to the roof: the lateral stresses dissipate by bending of the slab. In hard, solid, brittle rock the sidewalls normally

fail by slabbing, while in softer rocks (particularly in laminated shales or mudstones) the rock will flow into the opening reducing its cross-sectional area (see Fig. 8.8).

Fig. 8.9 shows the results of tests on the strength of different openings, done by Hobbs (1969; in: Thomas, 1978) in laminated plaster sand mixes. The trapezoidal opening was the weakest, and in order of increasing strength came rectangular openings, wide arches, narrow arches, circles of decreasing diameter, a pentagon, and then a hexagon with a vertical apex.

The depth of the underground workings determines the cover load, and the pillar dimensions must adjust to these increased compressive stresses. Fig. 8.10 shows the variation of the pillar dimensions and the extraction percentage with depth in British coal mines. The larger pillar dimensions in deeper workings reduces the influence of the seam width over the pillar stability: the strength of a 40 m² pillar is about the same whether the seam is, say, 1 m or 2 m thick.

thin immediate roof fails as a slender column

thick immediate roof tends to buckle and crush

in a wider room the lateral stresses are dissipated by bending
MINING UNDER SANDSTONE ROOFS

first heading caves, second remains intact
MINING UNDER SHALE ROOFS

thick sandstone
no deflection
even stress on pillar

thin sandstone bends
stress peaks especially at top edges
of pillar

PILLAR STRESSES

FIG.8.7 (Thomas,1978)

Roof failures at White Pine mine

FAILURE OF BRITTLE OR HARD ROCKS
BY SLABBING

FAILURE OF SOFT ROCKS BY FLOW
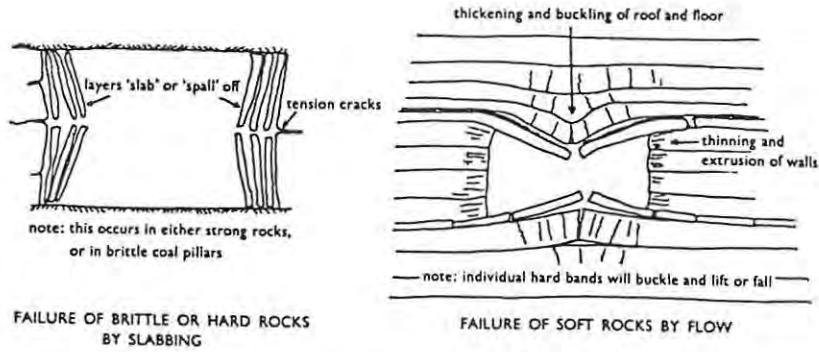
Failure pattern of roadways

FIG.8.8
(Thomas,1978)
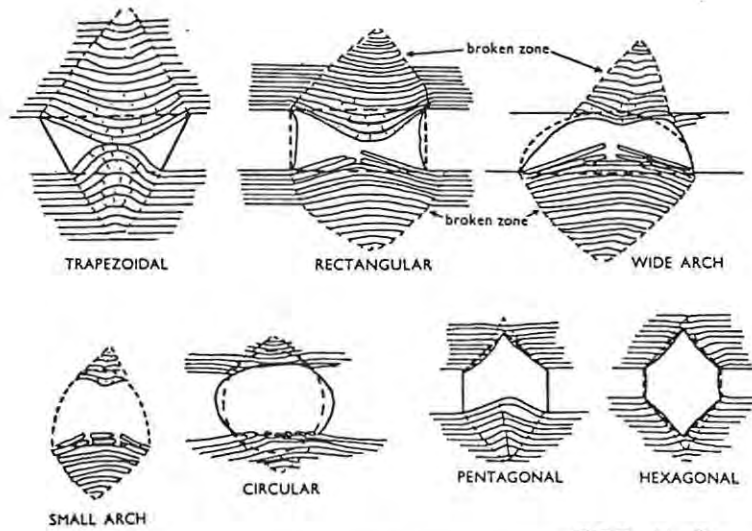
Roadway shape tests in bedded strata
(based on Hobbs[12])

FIG.8.9
(Thomas,1978)

Scale of pillars for first
working with the design of afterwards
taking out the pillars (M. Dunn)

Depth (m)	Dimensions of pillars (m)	Percentage extracted
36.5	18 × 4.5	59
73	18 × 5.5	50
110	20 × 6.5	48
146	20 × 7.25	43
183	20 × 8.25	41
219.5	20 × 9	39
256	22 × 10	37
292.5	22 × 11.75	34
329	24 × 17.75	31
366	24 × 14.5	30
402.5	25.5 × 16.5	27
439	25.5 × 18	24
476	27.5 × 19	23
512	27.5 × 20.5	20
549	27.5 × 22	18

FIG.8.10
(Roberts,1981)
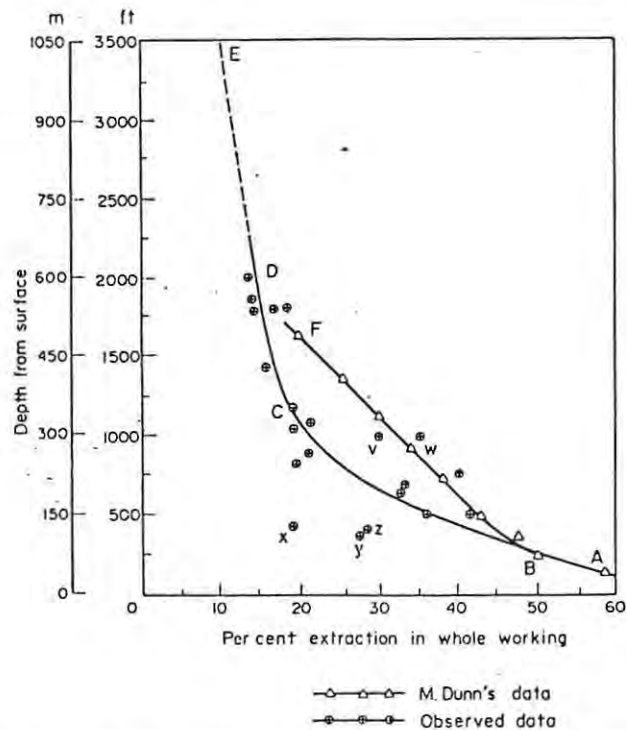
Variation of whole working extraction percentage with depth in British coal-mines.

As the span of the underground openings increases, an arch limiting rocks under tensional stress forms in the hangingwall. The process, and its distribution of stresses, is similar to the "arching" process described in Chapter 6 and illustrated in Fig. 6.1. When arching takes place, the stresses due to the cover load are distributed upon the sides of the opening, as shown in Fig. 8.11 for different situations.

FIG.8.11
(Thomas,1978)

Sequence of collapse in a mined area
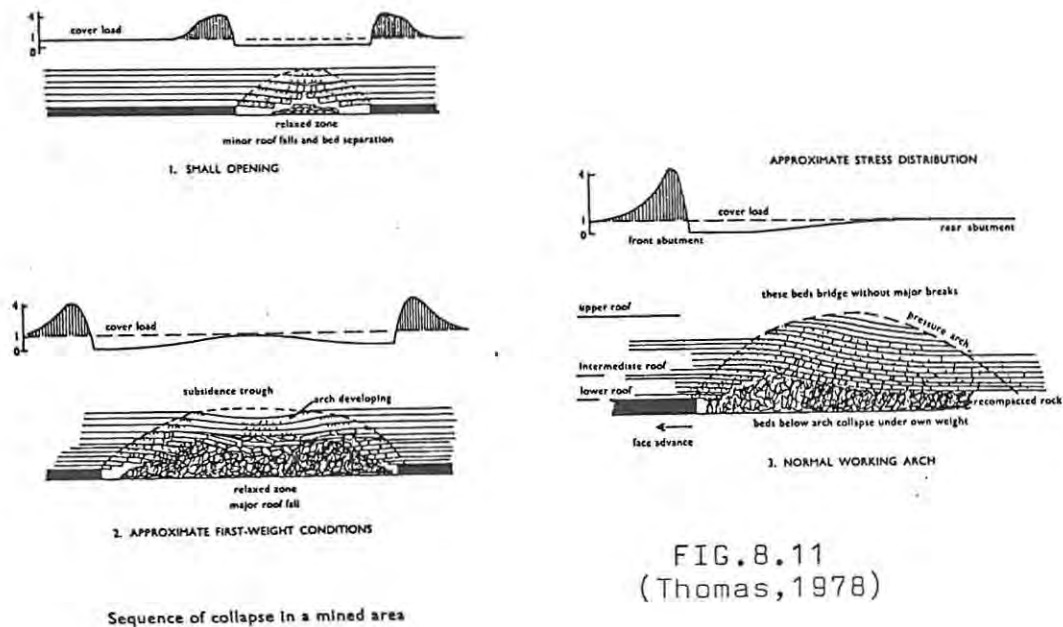
A room-and-pillar layout normally consists of several "panels" separated by barrier pillars. The load on individual pillars should be always smaller than the corresponding pillar strength, and the barrier pillars should take the abutment loads. The cover load should be transferred onto the barrier pillars in a process of arching, and the smaller pillars inside the panels would be required merely to maintain the integrity of the roof inside the barriers (they would support the rock-mass under tensional stress within the arch above the opening). Fig. 8.12 shows a relationship between depth and maximum width of the pressure arch.

Depth (m)	Width of maximum pressure arch (m)
122	40
183	50
245	60
305	70
306	80
427	90
488	100
549	110

FIG.8.12

(Roberts,1981)

The reduction on the load upon the pillars within the panel allows an increase in the extraction percentage. Fig. 8.13 shows the probable stress distribution across a rib or barrier pillar. Fig. 8.14 shows the way how the stress patterns interfere around a longwall stope (whose hangingwall is supposed to collapse once the place has been mined out).
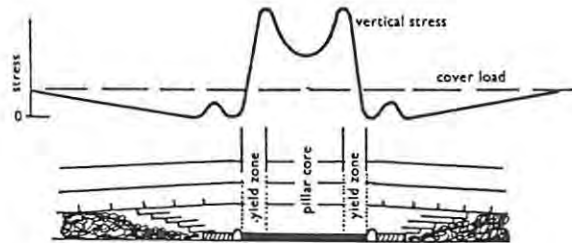
Probable stress distribution across a rib pillar
FIG.8.13 (Thomas,1978)

Room-and-pillar systems are most commonly laid out on a simple grid pattern, and the resulting pillars are square, rectangular, or more recently rhomboidal to adapt to less flexible equipment. In recent years, the Mining Research Centre of the Canadian Department of Energy, Mines and Resources has studied some alternative patterns that could result in radical changes in room-and-pillar layout design in the near future (Zahary, 1978).
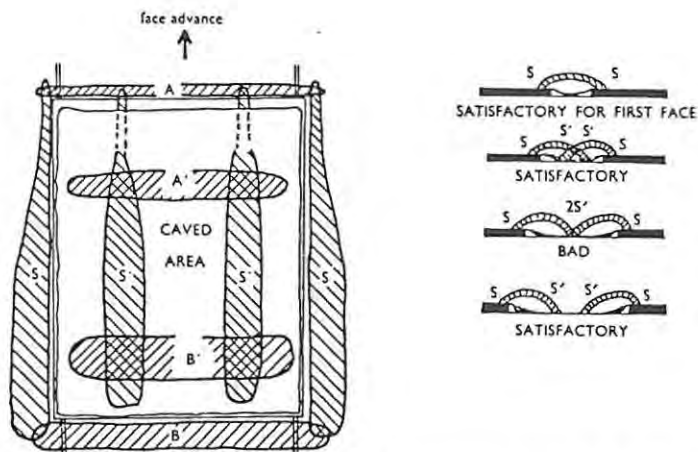
A = concentrated travelling stress zone in front abutment
B = distributed stress on rib where face was opened out
S = stress on side ribs redistributed in proportion to time and face advance

A' = distributed stress of travelling rear abutment
B' = distributed stress on caved waste after first weight collapses
S' = stress on caved waste redistributed in proportion to time and face advance

FIG.8.14 (Thomas,1978)
Interaction of stress patterns around an excavation

According to Zahary (1978), an optimum extraction policy is obtained when the room-and-pillar pattern meets three essential criteria:

(i) high pillar aspect ratio, defined as the ratio W/H, where the minimum pillar width W is the diameter of the largest circle that can be inscribed on the pillar's cross-section (H is the pillar height);

(ii) low absolute span of rooms and intersections, understanding as room span the driving width, and as intersection span the diameter of the circle inscribed in the intersections that is tangent to similar circles on the pillars; and

(iii) high support to span ratio, defined as the minimum width of the pillar divided by the maximum span of the intersections.

These criteria are said (Zahary, op. cit.) to be most effectively met when rooms are driven at 60° to each other, with only two rooms intersecting at one point, as shown in Fig. 8.15. The triangular pillars left are subsequently mined out (see Fig. 8.16).

In Figs. 8.17, 8.18 and 8.19 the resulting hexagonal (and hexagonal-triangular) pattern is compared with other pillar patterns. Fig. 8.17 shows that for a fixed extraction ratio of 50%, the hexagonal pattern gives the highest pillar aspect ratio (W/H). In Fig. 8.18, it can be seen that this pattern gives one of the highest (only exceeded by the trangular pattern) maximum intersection span to room width ratios, but if we consider the temporary support of the triangular secondary pillars then the hexagonal-triangular pattern results in the lowest intersection span. Fig. 8.19 shows that the hexagonal and hexagonal-triangular patterns results in the highest support to span ratios for any given extraction ratio.

The hexagonal pattern results in a more stable ground and/or in higher extraction percentage. It also makes possible to define mining sequences so that the sharpest corner in the mining/haulage route is no less than 120°, and the turn radius of LHD equipment and trucks is easily accommodated.

Hexagonal-triangular pillar pattern
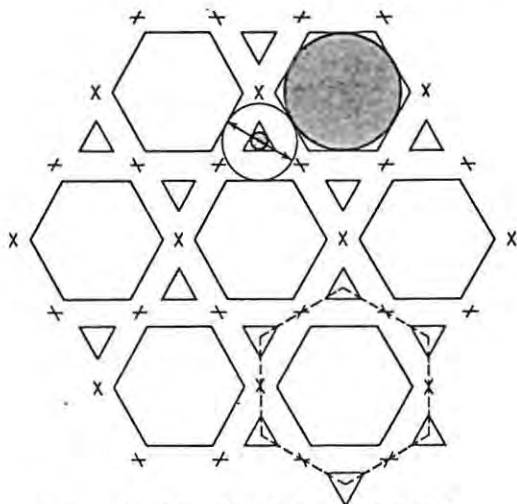
FIG.8.15 (Zahary,1978)

Hexagonal pillar pattern
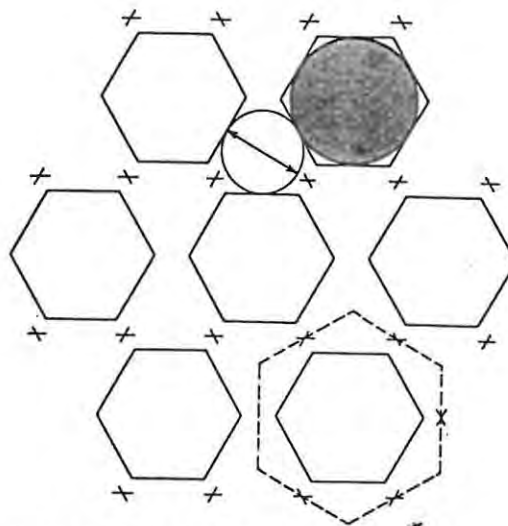
FIG.8.16 (Zahary,1978)

Variation of minimum pillar width with room width
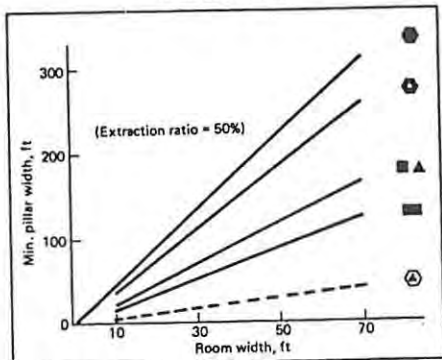
FIG.8.17 (Zahary,1978)

Variation of maximum intersection span FIG.8.18 **with room width**
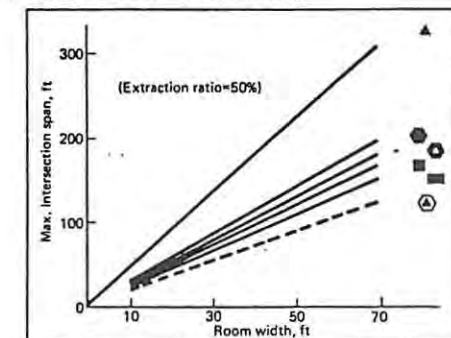
Variation of support/span ratio with extraction ratio
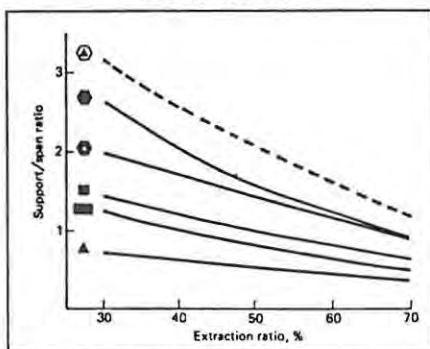
FIG.8.19 (Zahary,1978)

Various pillar patterns:

▲ Triangular

⬢ Hexagonal (hexagonal - triangular pattern)

△ Triangular (hexagonal - triangular pattern)

○ Hexagonal

■ Square

▬ Rectangular   FIG.8.20

8.3. Layout planning

The oldest form of room-and-pillar, which goes back to the beginning of underground mining itself, is one in which pillars are left in an irregular pattern, only where necessary and trying to locate them in zones of waste or lower values (see Fig. 8.21). Where the ore is regularly distributed in tabular, subhorizontal orebodies, as in coal, volcanosedimentary iron and manganese, trona, gilsonite, potash, oil shale, salt, limestone, and sandstone mines, the layout can be planned to the last detail, resulting in a uniform room-and-pillar mine. Most metalliferous mines using room-and-pillar try to mine as regular a pattern as possible, but changes in height, width, thickness, dip, and grade of the ore - as well as the presence of faults, dykes, etc., interrupting the continuity of the orebodies or affecting the stability of the openings - result in comparable changes in the mine plan.
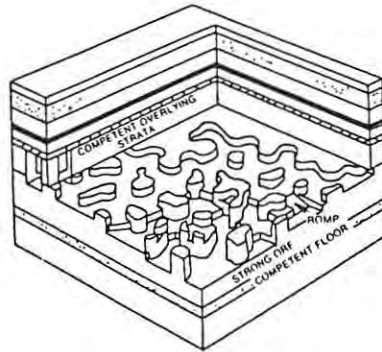
Open-stope rooms with random pillar supports (Anon., 1972).

FIG.8.21 (Bullock,1982)
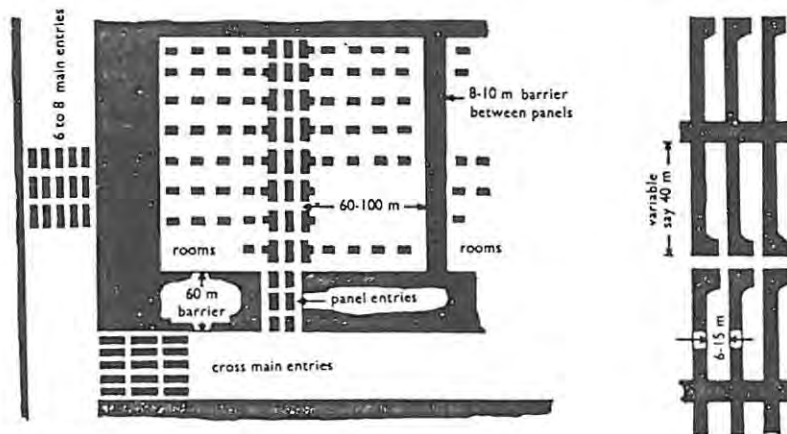
A. SHALLOW ROOM AND PILLAR WORKING   B. DEEPER BORD AND PILLAR LAYOUT

note: both these systems have been called 'Welsh Bords' in Australia

Room and pillar layout   FIG.8.22 (Thomas,1978)

Flat-dipping orebodies are frequently divided into 60 m to 120 m wide panels, separated by rib or barrier pillars, as shown in Fig. 8.22 for a coal example. The stresses of the cover load are distributed by arching upon the rib-pillars, in the way described in the previous section, and the smaller pillars within the panel will only take the weight of the roof rocks inside the arch that limits the mass under tensional stress. Fig. 8.12 shows some maximum widths of the pressure arch for different depth; in practice, the optimum panel width is determined by measuring the stress field and using photoelastic models. The rib pillars protect the permanent transport openings, and the pillars within the panel can be made smaller thus increasing the ore recovery: in South African gold, platinum and chromium mines these smaller pillars are normally replaced by wooden packs, that yield slightly in order to achieve the stress transfer by arching.

In moderately-dipping tabular orebodies the rib pillars separating the panels are orientated along the dip direction, which is the optimum orientation in inclined strata (see previous section). Horizontal transport drifts, following the strike of the orebodies, are spaced at distances determined by the production rate of the stope thus defined and the transport capacity on these drifts. In these stopes mining can proceed maintaining a horizontal working face that retreats towards the upper level, the "breast stoping" shown in Fig. 8.25, or a down-dip face that retreats from a slot raise (see Figs. 8.23 and 8.26). A more irregular layout and mining sequence is shown in Fig. 8.26; it mixes different blasting techniques. Mucking is normally done by scrapers (see, in particular, Fig. 8.24) arranged in different patterns: double-drum scrapers, hand lashing plus strike scrapers, dip and strike scrapers, etc.

Fig. 8.27 shows the way the room-and-pillar method has been adapted to mine inclined orebodies using trackless equipment (which cannot operate on gradients steeper than about 20%). The production rooms are horizontal and they follow the strike of the orebodies, and a transport ramp joins the different levels. The sequence of blasting shown in Fig. 8.27 is not the only one that can be used: at the Otjihase copper mine (SWA) the 11 m wide rooms, between 4 m wide pillars, are cut in two 5.5 m wide sub-horizontal drifts, but the blasting sequence is the opposite to that shown in Fig. 8.27 (the drifts 2 and 4 are cut first, and 1 and 3 are cut in a second stage).
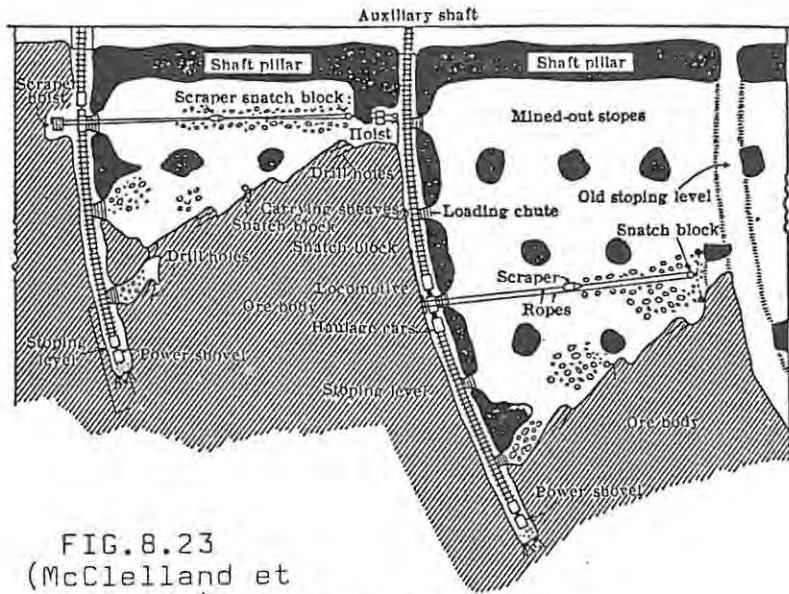
FIG.8.23
(McClelland et
al.,1941)

Breast Stoping, Mineville, N Y
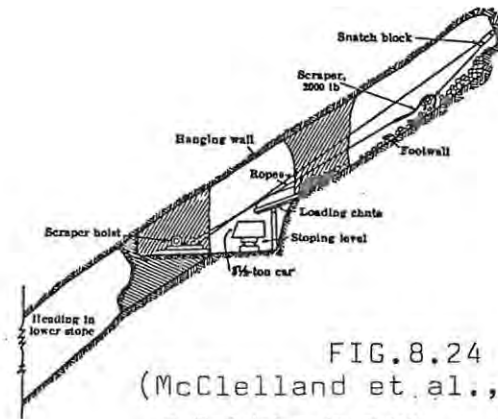
FIG.8.24
(McClelland et al.,1941)

Handling Ore by Scraper, Mineville, N Y
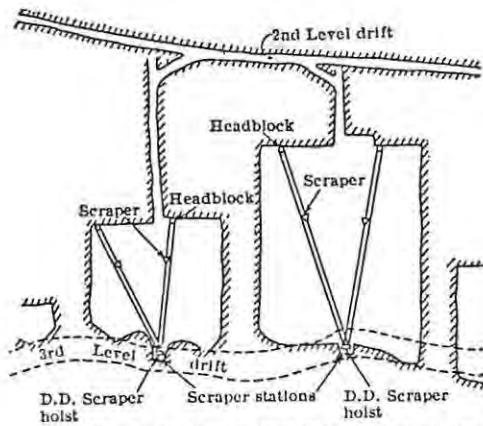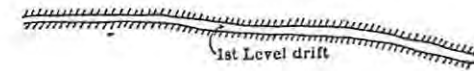
FIG.8.25
(McClelland
et al.,1941)

Breast Stoping in Flat-lying Section of Ore-
body, Sheritt Gordon Mine, Manitoba
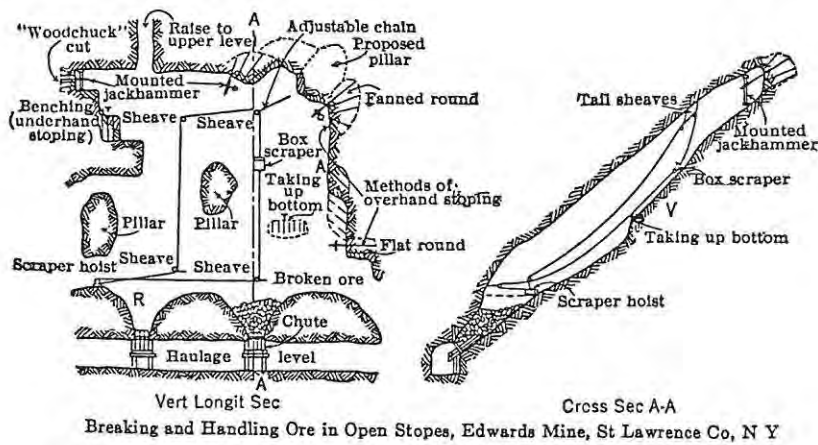
FIG.8.26
(McClelland
et al.,1941)

Breaking and Handling Ore in Open Stopes, Edwards Mine, St Lawrence Co, N Y
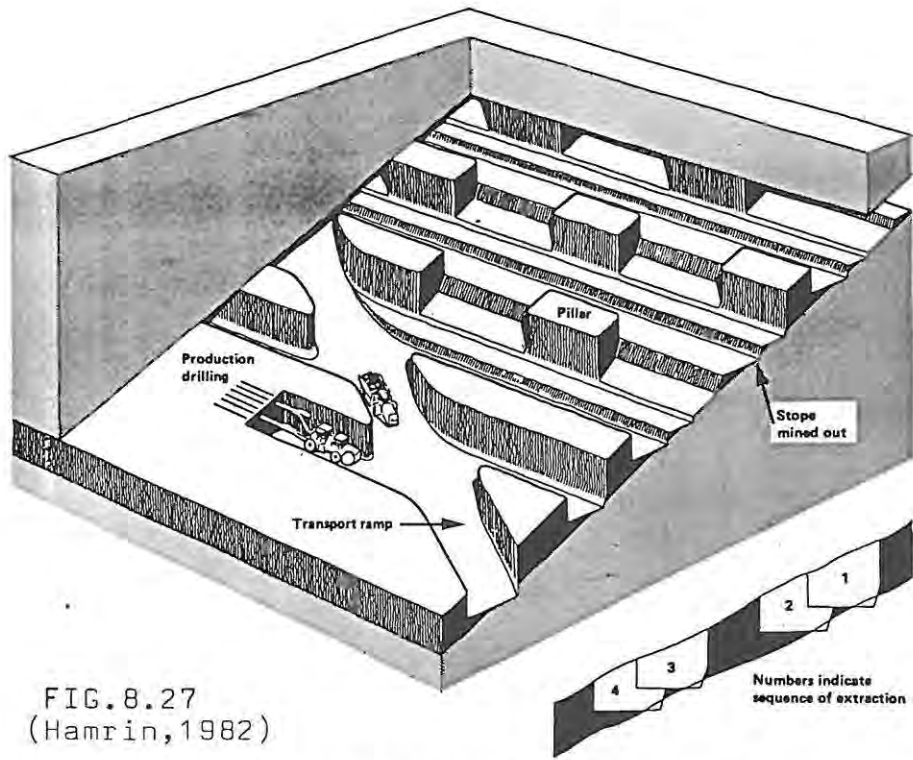
FIG.8.27
(Hamrin,1982)

Room-and-pillar mining, using step mining on an inclined ore body.

Fig. 8.28 shows the way the room-and-pillar layout has been adapted to the mining of the irregularly shaped pockets of lead-zinc mineralization at the Brushy Creek mine (E/MJ, 1978), a Mississippi Valley-type deposit in southeastern Missouri. The mine has not been divided into panels, since the sizes of the pockets are smaller than the normal panel size. The orientation of the rows of pillars tries to facilitate the movement

of the transport equipment (LHD units for short tramming distances, and
front-end loaders plus 40- to 50- ton trucks for long ones). The layout,
consisting of 7.6 m square pillars with a 10.7 m clearance, produces an
extraction ratio of roughly 82%, for this 4.3 m to 15.2 m thick sub-
horizontal, tabular orebody located at a depth of 300 m below surface.
Note in Fig. 8.28 the possibility of selective mining in a fairly regular
room-and-pillar layout: pockets of waste are left behind as pillars
(larger than the ones required by the regular mining pattern). The
workings should be orientated at right angles to faults, dykes or waste
bodies (that will make necessary to leave protective pillars or waste
pillars) so as to reduce the ore losses in pillars.
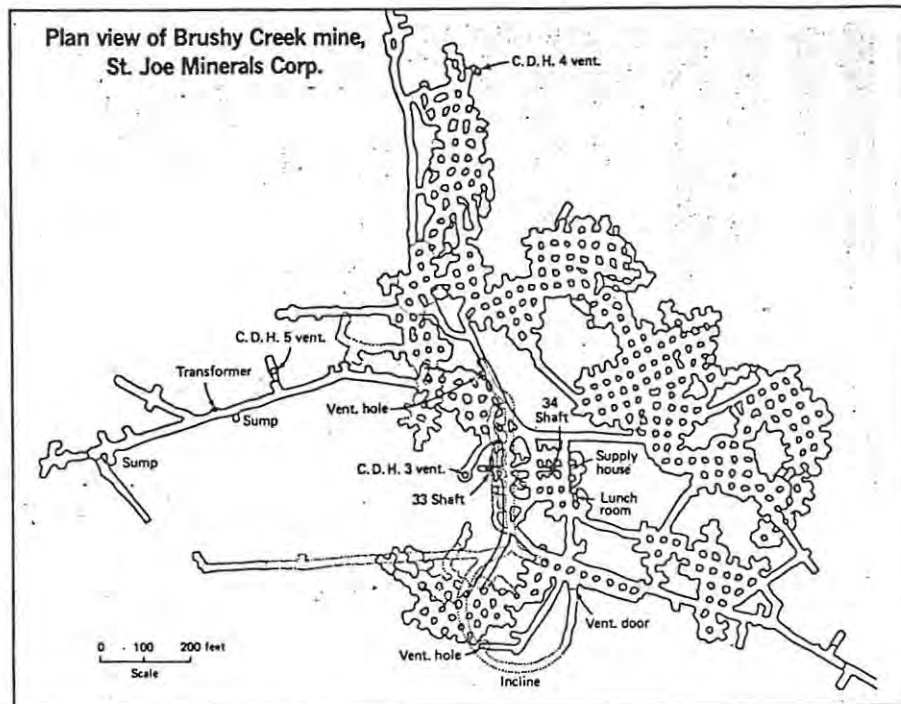
FIG.8.28 (Sisselman,ed.,1978)

When the orebody has not been subdivided into panels, and the
pillars are taking the cover load, the size of the openings at shallow
depths in moderately wide (2 to 5 m) flat-dipping orebodies is frequently
within the 6 m to 12 m range, and the pillar width within the 6 to 8 m
range. Under increasing load pressure conditions, the size of the
openings is reduced and that of the pillars increased, resulting in a
lower percent extraction that - for British coal mines - follows the
relationship in Fig. 8.10.

Due to the introduction of trackless equipment to underground mining operations, herringbone (see Fig. 8.29) and rhomboidal layouts (see Fig. 8.30) have been developed to accommodate the turn radius of the equipment.

Wide orebodies are mined in several slices, by benching (see Figs. 8.31 and 8.32) or by back-slabbing working on top of the muck pile (see Fig. 8.33). This type of mining allows a certain selectivity when mining an orebody consisting of layers of ore with different grades or compositions.

The accesses to the panels have three basic purposes: the mineral must come out, the men and materials must pass both ways, and sufficient ventilation must be provided. The third condition is particularly important in coal mining, where not only the minimum oxygen content of 19% must be met but also the amount of noxious or inflammable gases must be kept to a minimum; Fig. 8.34 shows the complex arrangement frequently used in coal mining to satisfy these requirements. Ventilation requirements for diesel equipment can be roughly estimated using the relationship: 100 cu. ft. per minute for every 1 hp of the diesel equipment in operation.

PANEL BELT

MAIN BELT

FIG.8.29 (Given,1973)

Development drifting at Black Pine mine
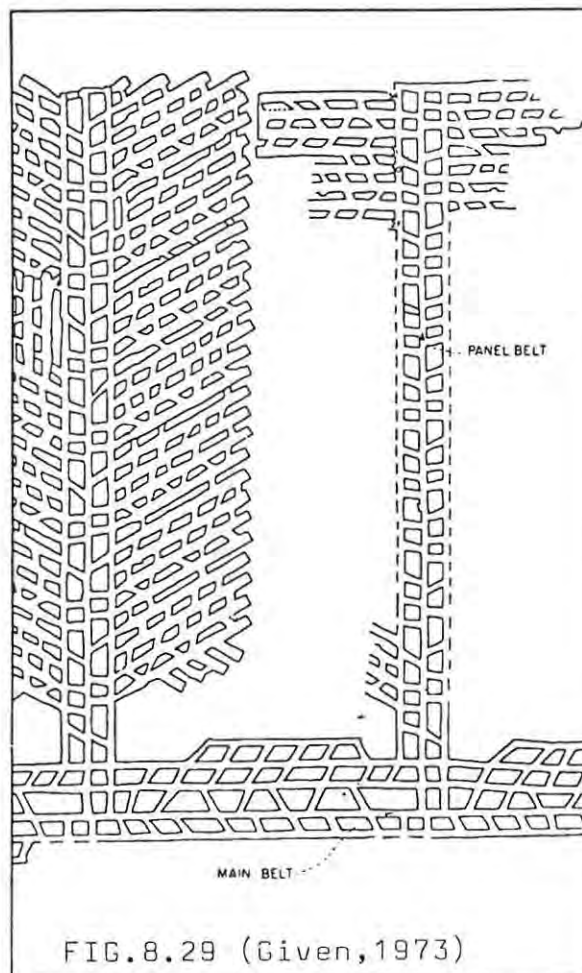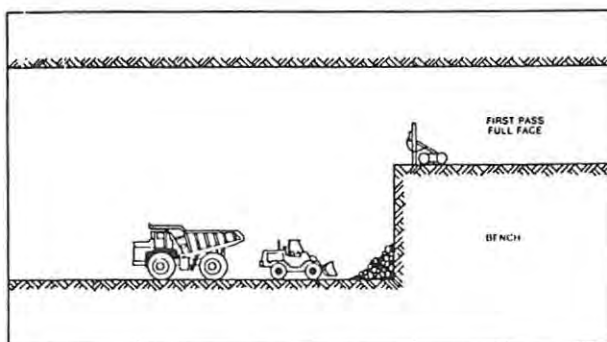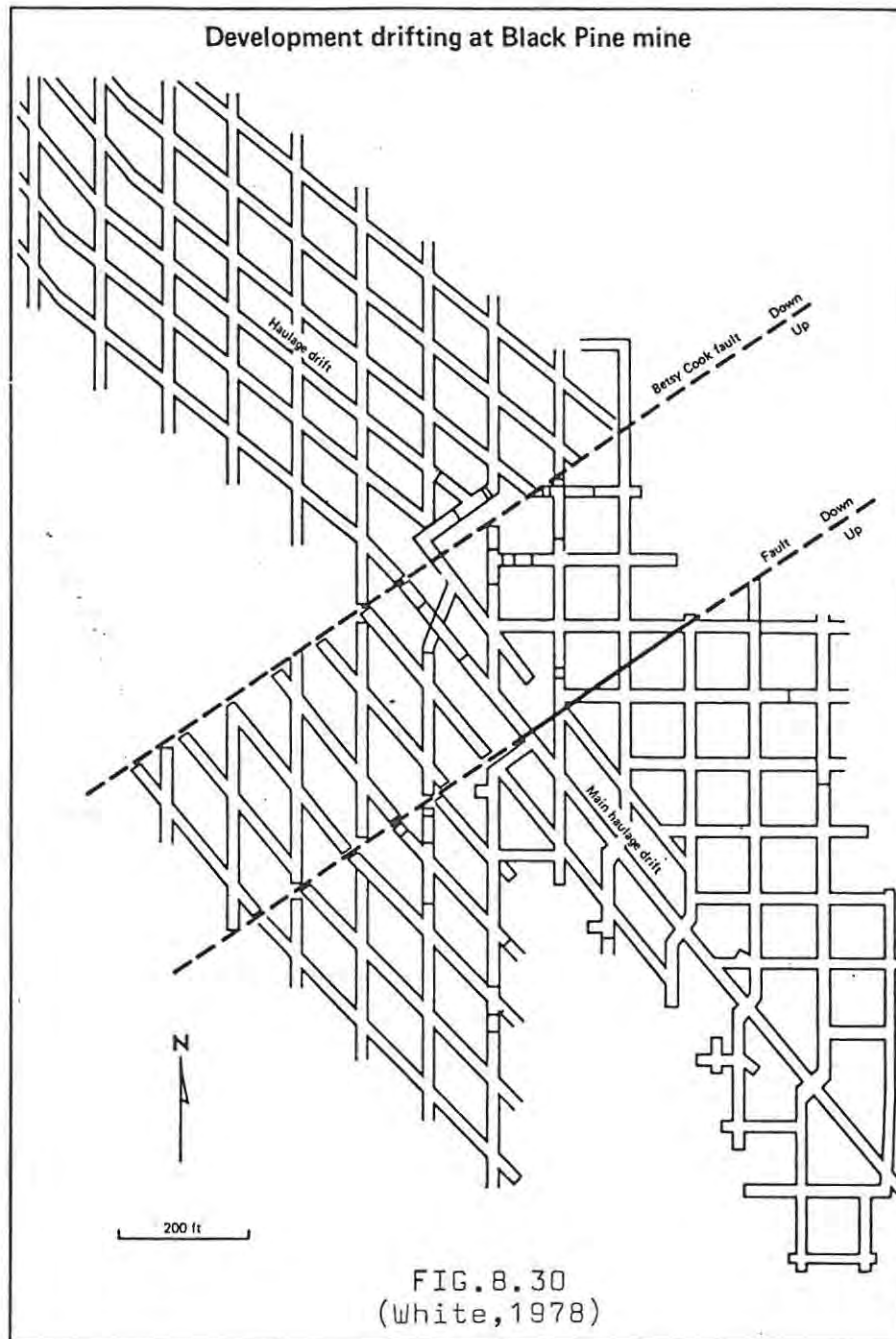
FIG.8.30
(White,1978)

Cross section through room of typical full face and
bench mining operation with the bench being blasted with
vertical drilling (after Dravo, 1974).

FIG.8.31 (Bullock,1982)
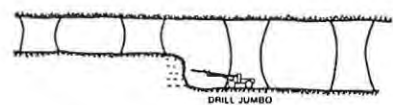
Cross section of stope show-
ing initial cut at top of ore, with
succeeding benches taken by hori-
zontal drilling (Casteel, 1973).

FIG.8.32 (Bullock,1982)
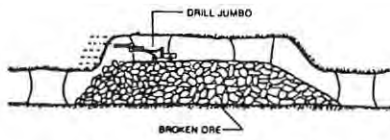
Cross section at bottom of stope showing initial cut at bottom of ore with remaining ore being removed by back-slabbing (Casteel, 1973).

FIG.8.33 (Bullock,1982)

Main entries plan
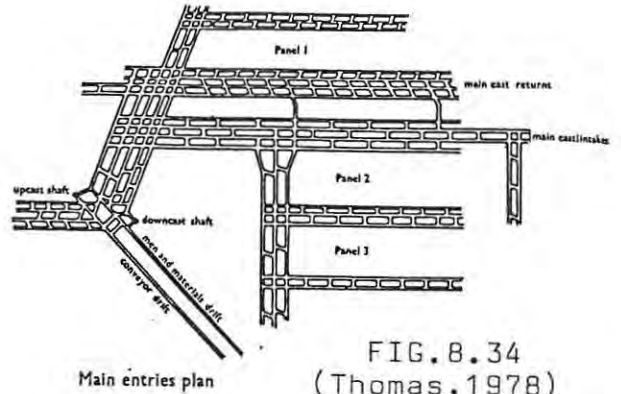
FIG.8.34 (Thomas,1978)
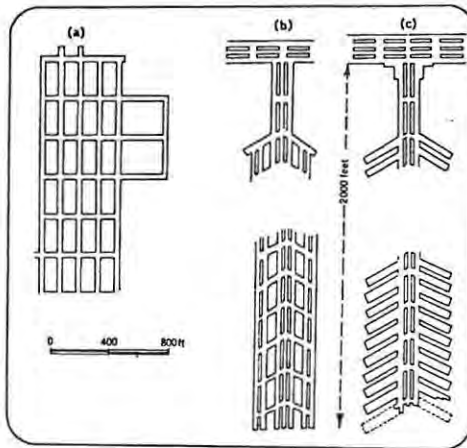
FIG.8.35 (Min.Mag., May,1978)

In hard ground sudden pillar collapse takes place when the elasticity limit of the pillars is reached before that of the hangingwall, but in soft ground the pillars collapse by creeping.   At the Cory potash mine in Saskatchewan (Mining Magazine, May, 1978), up to 7.6 m thick subhorizontal potash beds are being mined by room-and-pillar at a depth of roughly 1000 m. The original mining layout shown in Fig. 8.35 (a), which gave initial recovery ratios of 25%, has been replaced by the stress-relief and herringbone layouts shown in (b) and (c), respectively, in Fig. 8.35.   The multi-access entries layout looks similar to that in Fig. 8.34, but it serves different purposes.   Three parallel entries of approximately equal dimensions are driven along the main accesses and along the center of each panel.   The small pillars in between the main triple access roads are designed to fail, by creeping or plastic flow in the soft potash, thus relieving the stresses in the excavated area (the mechanism of stress relief is similar to the one applied in the yield-pillar technique, which is

discussed further on in this section). The centre entry remains open and stable for travelling way and conveyor installations. Pillars are also left between rooms to give initial roof support, and 122 m wide abutment barrier pillars are left between panels to take the cover load stresses by arching. The extraction ratio in panels of these new patterns varies from 15% to 50%, and the turn radius of trackless equipment is accommodated in a better way.

When surface subsidence and preservation of accesses to underground workings are not important, the hangingwall can be let to cave as mining retreats, in the way shown in Fig. 8.11. These techniques are frequently applied in coal mining, but they have been used in metalliferous mining as well. Fig. 8.36 shows a method of pillar cutting, removal and caving in use at the Mairy iron ore mine in France; this "fine-tuned" room-and-pillar is reported to yield iron ore at a cost of US$4 per ton (1978 dollars). When using these techniques, the effect of the front abutments on the mine openings must be taken into consideration. Fig. 8.37 (b) shows the yield-pillar technique : the idea is to have, interposed between the major abutment zones, a region in which the roof loads are carried by pillars too large to crush but weak enough to yield slightly, causing a transfer of the cover load on to the abutments; the recovery of those pillars becomes safer than when wide pillars are used and the stresses of the front abutment zone are distributed upon the workings (see Fig. 8.37 (a)). Ore recoveries are larger as well.

FIG.8.36 (Hoppe,1978)

The yield-pillar technique: (a) Pillars too wide to yield. The front abutment will damage the development openings. (b) Pillars narrow enough to yield. Main roof load is transferred to the abutments. The development area is narrow, so that it lies within the abutments (Woodruff [31]).

FIG.8.37   (Roberts,1981)

8.4. Evaluation and grade control

The main problem in ore reserve estimation and grade control in room-and-pillar operations is that of delineating the recoverable ore reserves, which involves optimizing the pillar support layout in order to obtain the maximum ore recovery for a given set of geological, rock mechanics, and mining equipment constraints.

At the Elliot Lake uranium area (Hart and Sprague, 1968), the 10° to 20° dipping unconformity-type uranium deposits have been mined using room-and-pillar, which consists of the two basic layouts shown in Fig. 8.38. The layout used at Nordic has been designed for shallow orebodies, generally located at depths of less than 1400' (427 m), and it gives 84% ore receovery. The Milliken layout has been used with orebodies at depths of 1900' to 3600' (579 m to 1097 m).

The sampling and estimation of ore reserves in an unconformity-type uranium deposit has got its own problems, derived from the irregular distribution of the values, but plans similar to the one in Fig. 8.39 (from the Elliot Lake field) can also be used to delineate and estimate recoverable ore reserves in other circumstances and with other mining layouts.   The section used can be a horizontal one, onto which the orebody, grade-intersections and mining layout are projected at right angles, or one parallel to the orebody.
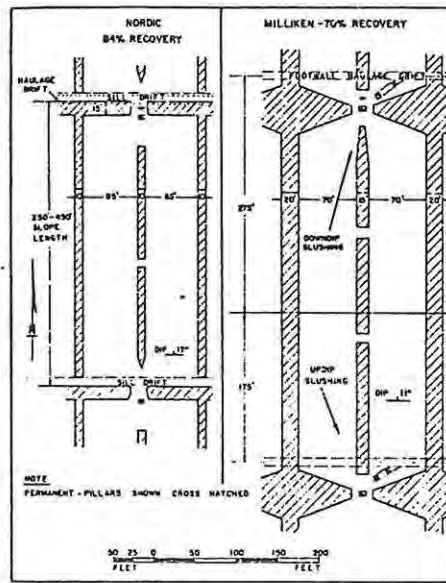
Typical Longwall Stope Plans for a 70-per-cent
and an 84-per-cent Recovery Factor.

FIG.8.38 (Hart & Sprague,
1968)
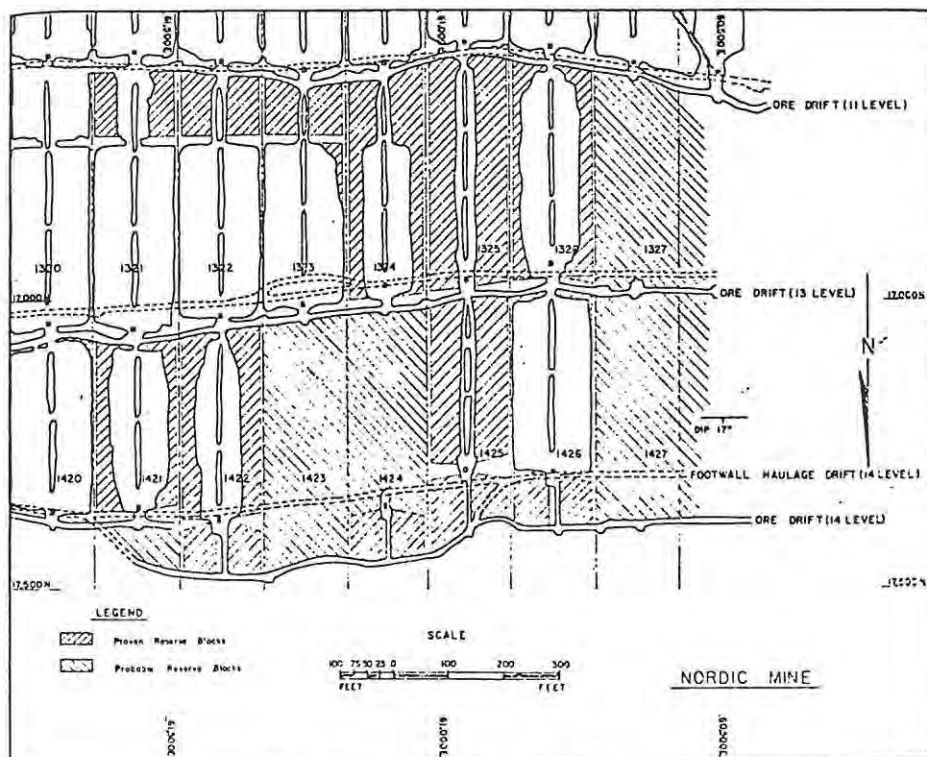
Typical Proven and Probable Ore Reserve Blocks.

FIG.8.39 (Hart & Sprague,1968)
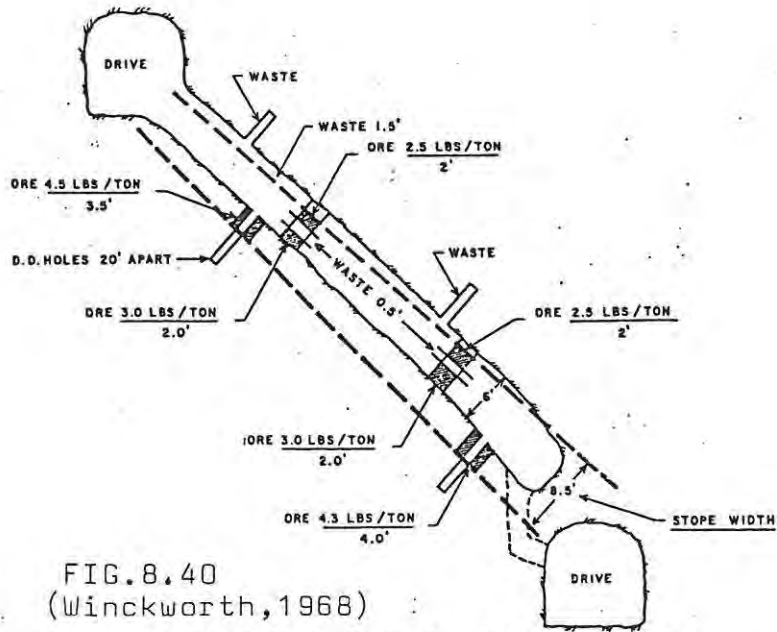
FIG.8.40
(Winckworth,1968)

Schematic Diagram Showing the Raise Section, with Raise and
Drill Values and the Subsequent Stope Width.
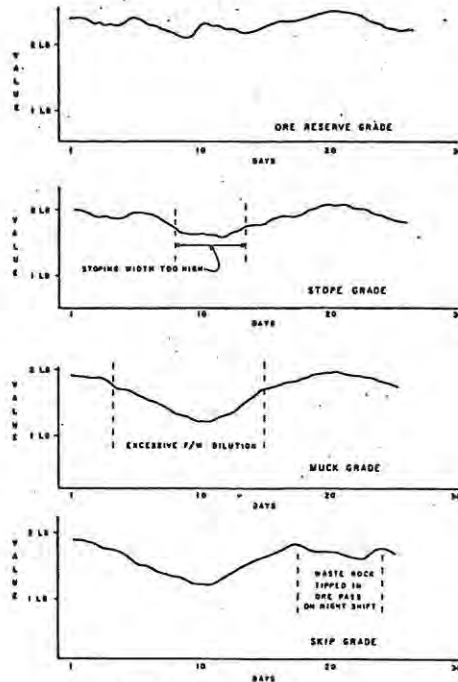
Graphs Showing Ore Reserve Grade, Stope
Grade, Muck Grade and Skip Grade.

FIG.8.41 (Winckworth,
1968)

(Vertical cross-sections can help to visualize the distribution of the ore across the orebody). Partial estimates for each block into which the orebody has been divided (each corresponding to a panel or mining unit) can be obtained using the polygonal method or grade×thickness contours on these sections, or, better, assigning geological areas of influence to the diamond drill intersections or any other sample plotted on the section. Estimates of proven ore reserves are normally based on the vertical exploration drill holes, and on channel or chip sampling of the access drifts and slot raise.

Not mining to the correct hanging and footwall contacts is a frequent grade control problem, and at the Rio Algom uranium properties in the Elliot Lake area (Winckworth, 1968) sludge sampling is used to control the hanging and footwalls (see Fig. 8.40). When the orebodies have "wavy" or gradational contacts, or when the contacts can only be determined by assaying, the dilution by overbreaking of hanging or footwall waste can be significant. In Fig. 8.41 a comparison among the predicted daily production grade (based on the ore reserve estimates and mining schedules), the grade of the actually mined portion of the ore reserves ("stope grade", based on the estimates as well), the muck grade, and the skip-sampling grade is shown. The dilution due to overbreaking of footwall waste is reflected on the stope, muck, and skip grades, while dilution due to "misplacement" (waste rock tipped into the ore pass on the night shift) is reflected in the skip grade.

Kriging has been normally used to estimate ore reserves in South African gold mines using room-and-pillar in their mining operations, and the literature on geostatistics contains abundant examples of this application.

8.5. Economics of room-and-pillar

The productivity in room-and-pillar operations is related - among other factors - to the room size and to the extraction percentage, as it can be seen in Fig. 8.42 for some American mines (Bullock, 1982). With wide rooms the extraction percentage and the face area increase and productivities of up to 94.4 tons per manshift are achieved. With small rooms, the face area decreases and the productivity can be as low as 14.4

tons per manshift (for these mechanized trackless room-and-pillar operations). According to Halls (1982), the productivities can range between 9.1 and 136.1 tons per manshift, and they average 12.7 tons per manshift.

Productivity Relationships Between Room Size and Productivity

Type of Mine	Face Area, m² (sq ft)	Extraction, %	Production Rate, t/d (stpd)	Productivity t (st) per manshift
Limestone	156 (1680)	71	2700–4500 (3000–5000)	94.4 (104.1)
Limestone	178 (1920)	75	900–2700 (1000–3000)	66.7 (73.5)
Limestone	139 (1500)	66	2700–4500 (3000–5000)	53.3 (58.8)
Limestone	47 (510)	55	2700–4500 (3000–5000)	30.5 (33.6)
Metal ore	49 (528)	80	4500–6400 (5000–7000)	55.2 (60.9)
Metal ore	45 (480)	78	4500–6400 (5000–7000)	29.5 (32.5)
Metal ore	47 (504)	84	2700–4500 (3000–5000)	21.3 (23.5)
Metal ore	31 (339)	85	10 000+ (12,000+)	14.4 (15.9)

FIG.8.42 (Bullock,1982)

R.C. Howard-Goldsmith in his review of the copper mining industry (Min. Mag., Feb. 1978, pp.111) quotes an average mining cost of $3.50 per ton for room-and-pillar operations, which seems to be a rather low value. Hoppe (1978) quotes a $4 per ton costs (1978 dollars) for what he considers a "fine-tuned" room-and-pillar operation at the Mairy iron ore mine (France). Halls (1982) gives the following range for mining costs in room-and-pillar operations: $1.28 to $4.96 per ton (1977 dollars), and an average cost of $2.26 per ton. The total mining cost of $8.03 per ton (1977 dollars) quoted by Halls, in the same paper, for a 3600 ton per day trackless room-and-pillar operation seems to be close to a more realistic average ($4 to $6 per ton). Taking this operation as an example, the distribution of the mining costs, and in particular of the supervision, labour and supply costs, can be seen in Figs. 8.43, 8.44, 8.45 and 8.46.

Mining Costs for a 3600-t/d (4000-stpd) Room-and-Pillar Operation, $/t ($ per st)

Item	1977 Labor		1977 Supplies		1977 Electric Power	1977 Depreciation	1977 Total
	Operation	Maintenance	Operation	Maintenance			
Development	0.160 (0.145)	0.040 (0.036)	0.468 (0.425)	0.062 (0.056)	0.008 (0.007)	0.065 (0.059)	0.802 (0.728)
Production	0.949 (0.861)	0.542 (0.492)	1.058 (0.960)	0.556 (0.504)	0.072 (0.065)	0.585 (0.531)	3.762 (3.413)
Hauling	0.203 (0.184)	0.051 (0.046)	0.022 (0.020)	0.037 (0.034)	0.050 (0.045)	0.085 (0.777)	0.448 (0.406)
Hoisting	0.137 (0.125)	0.120 (0.109)	—	0.071 (0.064)	0.173 (0.157)	0.098 (0.089)	0.600 (0.544)
Pumping	—	0.020 (0.018)	—	0.023 (0.021)	0.240 (0.218)	0.015 (0.014)	0.299 (0.271)
Ventilation	—	0.020 (0.018)	—	0.008 (0.007)	0.180 (0.163)	0.018 (0.016)	0.225 (0.204)
Gen. Mine	0.473 (0.429)	0.211 (0.191)	0.058 (0.053)	0.095 (0.086)	0.114 (0.103)	0.148 (0.134)	1.098 (0.996)
Supervision	0.799 (0.725)	—	—	—	—	—	0.799 (0.725)
Total	2.722 (2.469)	1.003 (0.910)	1.606 (1.457)	0.851 (0.772)	0.836 (0.758)	1.014 (0.920)	8.031 (7.286)

FIG.8.43 (Halls,1982)

Supervision and Labor Costs for a 3600-t/d (4000-stpd) Room-and-Pillar Operation; Based on Two Shifts Per Day, Five Days per Week

Supervision	No.	Hourly Rated	No.
Mine Superintendent	1	Developers	6
General Foremen	4	Production	28
Foremen—Production	4	Rail Haulage	6
Foremen—Development	2	Scale and Roof Bolt	6
Foremen—Service	2	Diamond Drillers	4
Foremen—Supply	2	Hoist	3
Foremen—Maintenance	4	Cagers	3
Chief Engineer	1	Nippers	3
Mining Engineers	2	Road Grading	2
Geologists	2	Trainees and Reliefs	10
Safety Officer	1	Lamps, Dry, Toolmen	7
Shaft Boss	1	Maintenance—Equipment	28
		Maintenance—Service Truck	2
		Maintenance—Surface Shops	5
		Maintenance—General Underground	12
Totals	**26**		**129**

Annual Cost of Supervision	$ 725,000
Annual Cost of Hourly Rated	2,652,640
Annual Total Cost	**$3,377,640**

Maintenance Labor Cost Split:
Development	4%
Production	54%
Hauling	5%
Hoisting	12%
Ventilation	2%
Pumping	2%
General Mine	21%

FIG.8.44
(Halls,1982)

Development Supply Costs for a 3600-t/d (4000-stpd) Room-and-Pillar Operation

Item	1977 Cost, $/m ($ per linear ft)	1977 Cost Per Year
Haulage and Ramp—686 m (2250 linear ft) per year		
Drilling 1.2 x 1.2 m (4 x 4 ft) pattern, 3.5 m (11.5 ft) steel for a 3.2 m (10.5 linear ft) advance; 60 holes 65.7 m per m (65.7 ft per linear ft advanced.)		
Drilling & blasting at $1.97 m ($0.60 per foot) drilled	129.33 (39.42)	
Mucking—31.8 t at $0.28 per t (35 st at $0.25 per st)	28.71 (8.75)	
Support—roof bolts & mesh on 1.2 m (4 ft) centers	127.30 (38.80)	
Ventilation pipe (reclaimed)	18.21 (5.55)	
Air and water pipe	38.02 (11.59)	
Miscellaneous—pipe hangers, chain, etc.	16.40 (5.00)	
Track (haulage only)	82.02 (25.00)	
Road bed—50.2 m³/linear m at $5.33/m³ (20 cu ft per linear ft at $4.00 per cu ft)	262.47 (80.00)	
Total	702.46 (214.11)	$481,748
Orepass—54.9 m (180 linear ft) per year		
Drilling 20 holes with 2.1 m (7 ft) steel for 1.8 m (6 ft) advance; 23.3 m/linear m (23.3 ft/linear ft.)		
Drilling and blasting at $1.97/m ($0.60 per ft) drilled	45.87 (13.98)	
Mucking—4.64 t at $0.28/t (5.12 st at $0.25 per st)	4.20 (1.28)	
Support—three sides, roof bolts	43.41 (13.23)	
Temporary staging	6.56 (2.00)	
Total	100.03 (30.49)	$ 5,488
Total Development Supply Cost		**$487,236**

FIG.8.45 (Halls,1982)

Production Mining Supplies

Item	1977 Cost, $/m ($ per ft)
Rooms are 9.1 m (30 ft) by 11.0 m (36 ft) high.	
Top—6.1 m (20 ft) carried full face	
There are 65 holes at 3.5 m (11.5 ft) for 3.2 m (10.5 linear ft) advance = 71.2 m/m (71.2 ft per linear ft)	
Bench—4.9 m (16 ft) with 1.2 x 1.2 m (4 x 4 ft) hole spacing = 32.8 m/m (32.8 ft per linear ft)	
Total 104.0 m/m (104.0 ft per linear ft)	
Drilling and Blasting—104 m (104 ft) at $1.48 per m ($0.45 per ft)	153.54 (46.80)
Mucking—257.2 t/m (86.4 st per linear ft) at 0.15/t ($0.14 per st)	39.70 (12.10)
(Spares $14 per operating hour—90.7 mt/h (100 tph)	
Support—50% of roof; roof bolts at 1.2 m (4 ft) spacing	40.72 (12.41)
Total per linear m (ft)	233.96 ($71.31)
Total per mt (st) of ore	0.91 ($0.83)
Fuel and Lubrication	
There are 39 diesel-operated units underground, including LHDs, jumbos, graders, trucks, ANFO trucks, giraffes, and combination personnel-supply carriers. On a two-shift basis, five days per week, the annual operating machine hours would be approximately 40,000 at an approximate weighted average of $4 per hour.	
Annual fuel and lube ($15,000 charged to development)	$160,000
Surface vehicles	
Fuel and Lubrication	$ 30,000
Diamond Drill—1524 m (5000 ft) at $14.76/m ($4.50 per ft)	$ 22,500

FIG.8.46 (Halls,1982)

9. SHRINKAGE

9.1. The application of shrinkage

In shrinkage stoping a fraction (roughly 40%) of the broken ore is drawn by gravity to account for the swell, and the rest is used as a working platform to continue the overhand mining retreating in an upwards direction, breaking successive horizontal slices of ore.

Since the ore is drawn by gravity, the orebodies must be steep (normally over 60°). As dip falls below 70°, the shrinkage draw strongly favours the hangingwall side, thus leaving a poor working platform for overhand work (especially in relatively wide stopes). The support the fill of broken ore gives to the hangingwall also diminishes with the decreasing dip, being nil as the dip approaches the repose angle of that broken ore.

The minimum stoping width is that required for working space, which is about 1.2 to 1.5 m; orebodies narrower than that should carry the dilution of the waste to be mined in order to reach that minimum width. However, narrow stopes, close to the minimum width, are affected by frequent hang-ups and bridging in the broken ore, and the draw becomes erratical and the broken ore is not recovered completely; in these stopes, the stope height must be considerably reduced. Regularity of the orebodies along the dip is a prerequisite of shrinkage, as there must be no serious obstruction to the gravity flow of broken ore. When the orebodies locally narrow or pinch-out along dip, a sublevel and a new draw system must be developed or, alternatively, ore passes must be driven in the waste to connect the upper and lower portions in which the stope has been divided, hence resulting in either increased development costs or in draw problems. Gentle rolls along the dip of the orebody are acceptable if the local footwall dip everywhere exceeds 50°.

The maximum stoping width may be 3 m or less to over 30 m, depending upon the competency of the ore and its ability to stand unsupported across the stope back. Very wide veins and massive orebodies have been mined by transverse vertical shrinkage panels, separated by transverse vertical pillars which are either abandoned or recovered later by other methods

(mainly pillar blasting). The wall rock must be strong enough to stand with the support provided by the broken ore filling the stope, and slabbing of the wall rock during the final draw must be minimal in order to reduce the dilution.

Some physical and/or mineralogical characteristics of the broken ore may impose restrictions on the use of shrinkage stoping. Ores which, when broken, are cohesive or which tend to pack or cement together under the influence of ground water, wall pressure, or chemical reaction, produce an erratic draw during mining and often result in a difficult and/or incomplete final draw. Pyritic and pentlandite-rich ores which oxidize very rapidily in the stopes may generate heat, imposing a fire hazard by spontaneous combustion. Sulphide ores that oxidize sufficiently in the stopes may have a lower mill recovery by flotation. Ores (especially those containing uranium minerals) which exude radon gas impose ventilation constraints on stope design. In most cases these problems can be minimized by limiting the size of the stopes, and accelerating the final draw of the broken ore.

The relative advantages and disadvantages of shrinkage must be considered in comparing the method to alternative methods applicable to steeply-dipping tabular orebodies: sublevel stoping and cut-and-fill.

Compared to sublevel stoping, conventional shrinkage is a higher cost method, requires more competent ore for a given stope width, is more skilled labour intensive, is less easily mechanized, and has a poorer safety record. However, it will accommodate significantly weaker wall rock with less dilution, requires a smaller capital outlay and less development work, is generally more selective, and is applicable to veins or massive orebodies having a greater degree of irregularity or being smaller in size. Blasthole (or crater) and sublevel shrinkage, or variations thereof, are relatively new but promise to achieve the sublevel advantages of low cost, easy mechanization, safety and applicability to relatively weaker ores while retaining most of the shrinkage advantages. These gains are at the expense of whatever selectivity and ability to handle irregularities that conventional shrinkage has.

Compared to cut-and-fill stoping, conventional shrinkage is less expensive, generally requires less development work, and does not require the often elaborate systems for contemporaneous filling. Cut-and-fill stoping can, however, accommodate weaker wall rock and slightly weaker ore, has greater potential for mechanization, is more selective and flexible in its ability to accommodate irregularities in the orebody or walls, and generally gives better recovery with less dilution.

9.2. Layout planning

Shrinkage stopes are overhand stopes in which the broken ore fills the stope and is used for a working platform and to support the walls of the stope. As broken ore "swells" and requires a larger volume than the solid, some muck must be drawn off as the stope advances. (Hence the name "shrinkage"). Mining retreats in an upwards direction blasting down horizontal slices of the roof ore, and when mining ends the broken ore accumulated in the stope is drawn. Because gravity flow of the broken ore must be achieved, and the walls must remain stable during the mining and final draw of the ore, the method is best suited to narrow, steeply-dipping, vein-like orebodies.

The increase in the height of the stopes reduces the cost of developments per ton of ore, but, because approximately 60% of the broken ore remains in the stope until it is completed, the interest cost for the capital spent on developing and breaking this ore increases as well. Stope heights of 30 m or less to over 200 m have been used in the past, but 40 m to 60 m seems to be an acceptable range for stope heights; in any case, stope heights are seldom more than 75 m to 100 m. Stopes over 60 m to 90 m high tend to draw erratically; generally, flat-dipping, narrow or irregular veins require a close level spacing to minimize the problems of erratic draw. In deep mines or in stopes with weak sidewalls, the height must be reduced to decrease the dilution by slabbing of the sidewalls during the final draw; in narrow stopes between schistose sidewalls in gold mines of the Barberton area (South Africa), slabs of sidewall waste can completely obstruct the flow of broken ore.

The stope width must be such that the back of the stope remains stable during the stoping stage, and there is a minimum slabbing of the sidewalls (and dilution of the ore) during the final draw. Normally, the maximum stope widths are in the 6 m to 8 m range, exceptionally reaching over 30 m. The minimum stoping width is 1.2 to 1.5 m; it must provide enough room to work. In narrow stopes, hang-ups and bridging in the broken ore are frequent, resulting in an erratic draw and incomplete recovery of the broken ore. Wider orebodies could eventually be mined by a set of parallel shrinkage stopes, followed by blasting of the rib pillars once the stopes have been emptied, but other methods are preferred (methods with lower costs, higher productivities, and less dilution, which particularly affects the recovery of the rib pillars).

The main limitation to long stopes is, again, the delay in the start of the final draw and the increase in interest costs. Long stopes make difficult the access to and the ventilation of the stopes. Generally, 60 m to 90 m are frequent stope lengths.

Four to eight meters thick bottom and crown pillars are normally left, unless the use of loaders instead of boxes or scrapers eliminates the need for a bottom sill pillar. The crown pillar protects the haulage level above the stope, but if the right general mining sequence were followed (i.e. mining blocks or stopes from the upper levels down and retreating towards the shaft) it should not be necessary to preserve that haulage above by the time the mining of the stope reaches the crown pillar; in this situation the crown pillar provides support to the stope walls, thus reducing the dilution of the broken ore by caved waste, and it should be blasted once the final draw of the ore has concluded.

The first draw system that was used in shrinkage layouts consisted of a footwall transport drift driven in the orebody and raises, on the footwall side, to a "coning" or undercut level above the bottom pillar (see Fig. 9.1). The raises were subsequently sliped open ("coned") at the coning level, and boxes (chutes) were used to load cars running on tracks in the footwall drift. These drawpoints were normally spaced at 5 m or 6 m intervals, and the bottom sill pillar left above the footwall

drift was 4 m to 6 m thick.   A grizzly level was not necessary because shrinkage permits secondary blasting inside the stope, immediately after blasting down the corresponding slice (nevertheless, a grizzly is shown in one of the stopes in Fig. 9.2, which shows some shrinkage layouts used at the Kerr-Addison gold mines in Canada).   Alternatively, scrapers (slushers) have been used, instead of boxes and cars on tracks, in the footwall drift moving the muck from the drawpoints to a main ore pass (this system is frequently not very efficient and the draw lacks flexibility);   an example from the Crean Hill mine is shown in Fig. 9.3.
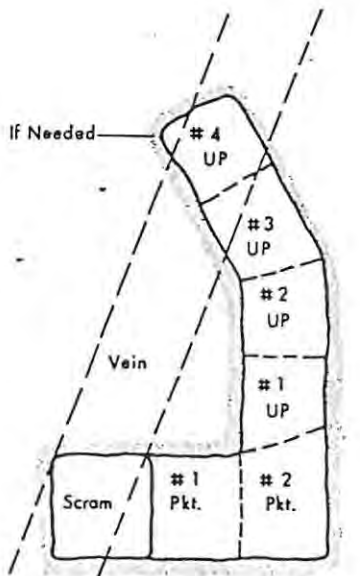
Cross section of hogging to catch the vein.

FIG.9.1 .(Hustrulid 1982)

HIGH SILL SHRINKAGE STOPE

LOW SILL SHRINKAGE STOPE
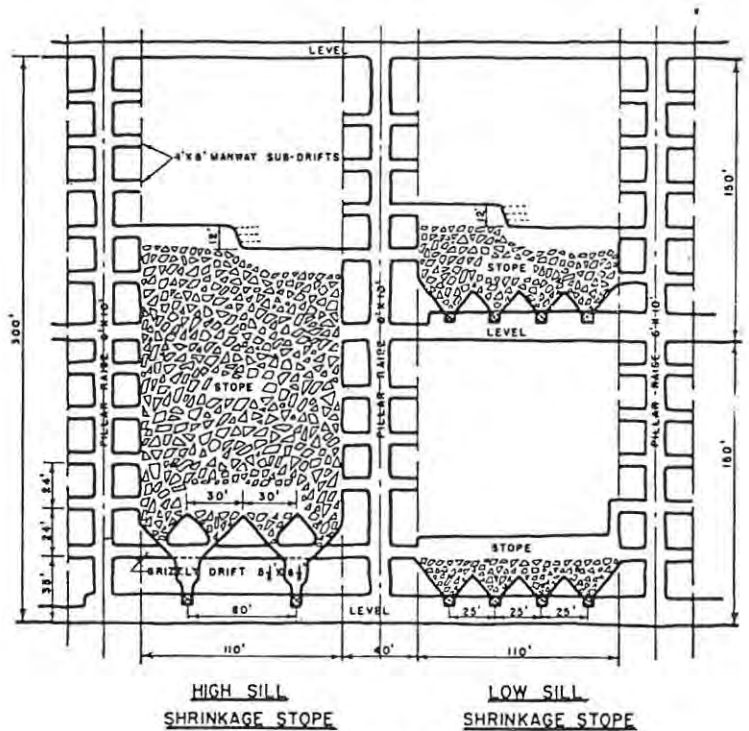
Elevation showing high and low sill shrinkage stopes.

FIG.9.2   (The Staff,1957)

The introduction of overhead loaders first and LHD units later changed the layouts of the draw systems.   The bottom pillar was eliminated and the boxes as well.   In narrow orebodies the footwall haulage drift is developed in the footwall waste and crosscuts are driven to the drawpoints.   When using overhead loaders on tracks (LM36 or similar) the crosscuts are about 5 m long, developed at right angles
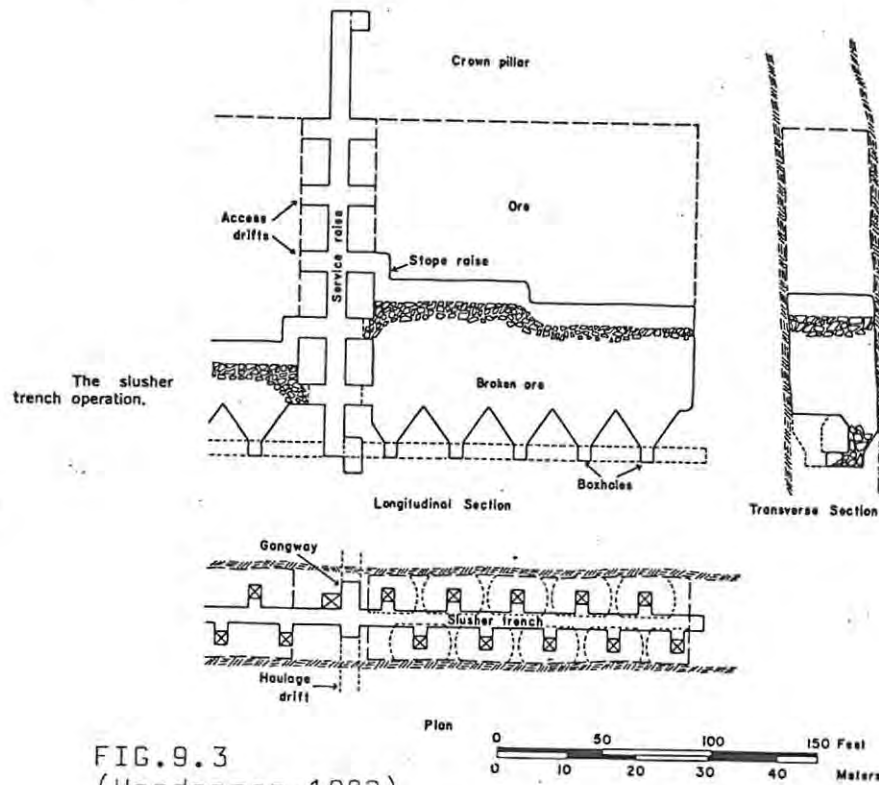
FIG.9.3
(Henderson,1982)

to the haulage, and spaced at 5 m to 8 m intervals along the haulage
drift (see Fig. 9.4).   When using LHD units the crosscuts are driven
at 60° angles to the haulage to accommodate the turn radium of the
equipment, their length increases up to 10 or 15 m, and their spacing
up to 15 m (see Fig. 9.5).
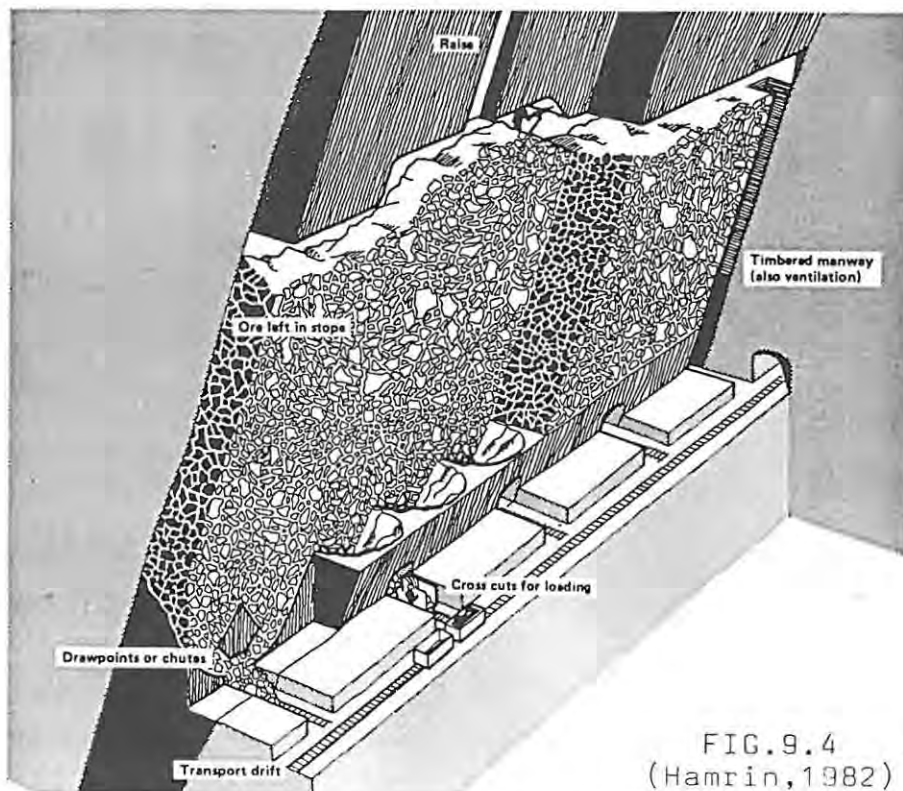
FIG.9.4
(Hamrin,1982)

Shrinkage stoping in a large vertical ore body.
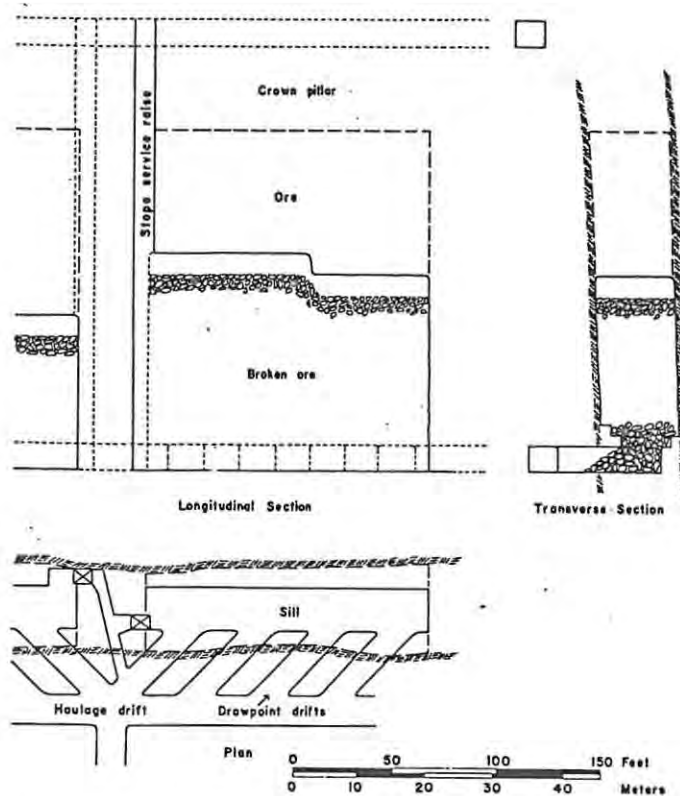
The load-haul dump (LHD) operation.

FIG.9.5 (Henderson, 1982)

When the orebody is about 9 m wide, all the developments of the draw system can be driven in ore. The haulage drift is driven in ore along the hangingwall contact of the orebody, and the crosscuts (in an angle to the transport drift, if LHD equipment is being used) go to drawpoints near the footwall contact of the orebody. This configuration results in stable brows at the drawpoints and in good gravity flow of the ore.

The draw system for wider orebodies consists of a haulage drift on the center of the orebody with cross cuts to drawpoints on both sides, or alternatively two haulage drifts: one in the footwall waste as previously described and a similar one on the hangingwall side. Fig. 9.6 shows a drawpoint design for overhead loaders in a wide (roughly 12 m wide) orebody. In a similar design for LHD units the crosscuts are laid out in a staggered pattern to optimize the draw and to reduce the span of the footwall openings, and in an angle to increase the length of the crosscuts to accommodate the equipment (staggered herringbone pattern). However, the pattern in Fig. 9.6 is the best for overhead loaders on tracks, loading into cars that run on tracks as well; the crosscuts have a smaller section and do not need to be in an angle to

SECTION A-A   SECTION B-B   SECTION C-C
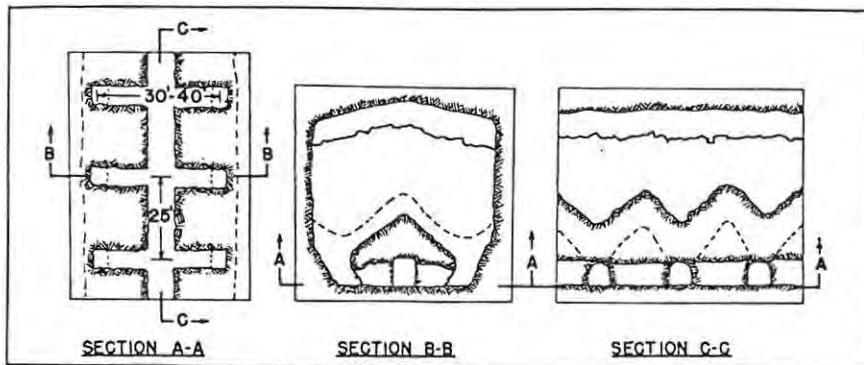
FIG.9.6
(Hoover,1973)

Drawpoints on haulage level in a large ore body.

the haulage, and laying them out opposite to each other reduces the
number of turn-tables (or, less likely, rail switches), hence reducing
the costs and making the tramming along the central drift easier and
faster.

The use of loaders and LHD units has increased considerably the
rate of draw and loading (and its productivity), has made possible to
handle larger fragment sizes, and has made easier to eliminate hang-ups.
Probably, the overhead loaders on tracks (LM36 or LM56-type) are the
best suited to the rates of production and the stope sizes of shrinkage.

Stope access and ventilation must be provided for conventional
overhand shrinkage although not for "blasthole shrinkage" (a variety of
shrinkage that uses the blasting techniques of vertical crater retreat :
this technique is much less selective and less flexible to follow
irregularities).   The most commonly used development is that shown in
Figs. 9.2 and 9.3.   A raise is driven or bored level-to-level (or
spanning several levels) on the orebody and at the extreme (s) of the
stope.   The raise is protected by rib pillars and short "doghole"
drifts are driven in both directions from the raise on the strike at
regular vertical intervals (typically 5 to 10 m).   Each raise serves
two stopes.   Entry to the stope is through the dogholes in ascending
order as the stope is advanced, each successive doghole being tapped by
a short raise driven from the stope back, just prior to the preceding
doghole being choked off with stope muck.   This short raise serves as
slot for the blasting of the next slice of roof ore.   Two vertical rib
pillars in ore remain after the stopes are completed, but they can be

drilled from the manway and recovered in a single blast after the stope has been emptied and the manway stripped.

Overhand stoping can be advanced by maintaining a flat back and blasting vertical or inclined (+ 70°) upholes, or horizontal holes, blasting down approximately 3 m wide roof slices at a time. Horizontal holes produce less damage to the back of the stope, but inclined blast-holes permit larger rates of production : a long extension of the stope back can be drilled and blasted in each round.

9.3. Evaluation and grade control

Ore reserves recoverable by shrinkage mining are evaluated in a way similar to that described for sublevel stoping : diamond drilling in cross-sections and use of the cross-sectional method of ore reserve estimation.

The method permits leaving behind pockets of waste as pillars, as shown in Fig. 9.7 (see pillar A). However, these pillars tend to obstruct the gravity flow of the broken ore and reduce the ore recovery. Some broken ore is lost between the drawpoints as well (see Fig. 9.8), and it normally corresponds to that blasted in the first cut that did not flow because of the drawpoint configuration. Off-shoots of the main orebody can be mined as long as gravity flow of their broken ore can be achieved; very rich off-shoots orientated at angles that impede gravity flow, may be mined using additional ore passes, or scraper or wheelbarrow mucking inside the stope, having sometimes to barricade off the openings after the mucking to avoid the overflow and losses of broken ore in them. Vertical offsets or major rolls along the dip which cannot be "smoothed over" generally require that a sublevel be established with new draw system development.

FIG.9.7
(McClelland et al.,1941)

Shrinkage Stope in Nevada-Massachusetts Mine, Mill City, Nev
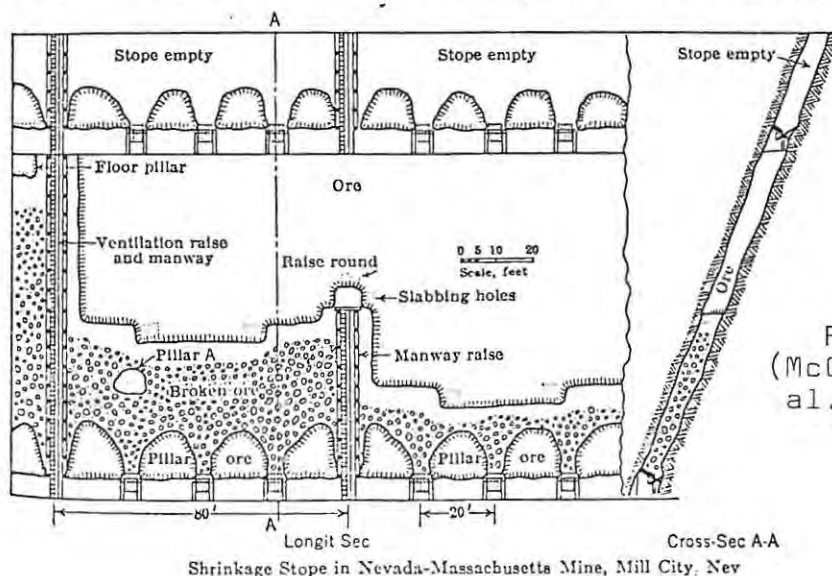
Dilution by overbreaking during the stoping stage is minimal:
0.3 m of overbreaking on each sidewall is noramlly the maximum when
the operation has a good supervision.   Dilution takes place mostly
during the final draw as a result of slabbing of the walls, but a good
draw practice can considerably reduce this type of dilution.   An
acceptable dilution in shrinkage mining is lower than 15%.   Ore
recoveries are roughly 70% with no pillar recovery, and they increase
to over 90% when the pillars are recovered.

Fig. 9.8 illustrates two methods of drawing down a shrinkage
stope after mining has been completed.   The method A consists in
drawing down from the end drawpoints first.   In this way the central
or weakest part of the stope wall is supported by the broken ore for
the longest possible period, and any wall-rock contamination rolls
down the ends of the pile and gathers in the empty drawpoints.
The method B consists in drawing down uniformly from all drawpoints.
In this case any waste caved from the walls accumulates in the funnel
openings above the drawpoints, and mixes with the broken ore because
of the "funnelling" in the ore (discussed in the chapter on caving),
thereby causing dilution.   The method A was successfully applied at
the Golden Manitou mines in Quebec (Hopper and Moore, 1957), reducing
the dilution to less than 15% in stopes with fairly weak sidewalls;
when the technique was used and at the same time the walls were roof-
bolted during the stoping stage, the dilution was reduced to less than
8%.

A good fragmentation, with the ore breaking to sizes smaller than
those of the caved waste, helps to reduce the dilution.   The knowledge
of the grades of the material broken during the stoping stage, makes
easy to schedule the drawing and to produce daily ore blends that suit
any requirement.

Diagrammatic sketch showing two methods of pulling a shrinkage stope.

A-1   B-1

A-2   B-2
FIRST STAGES OF MUCKING

A-3   B-3
FINAL STAGE OF MUCKING
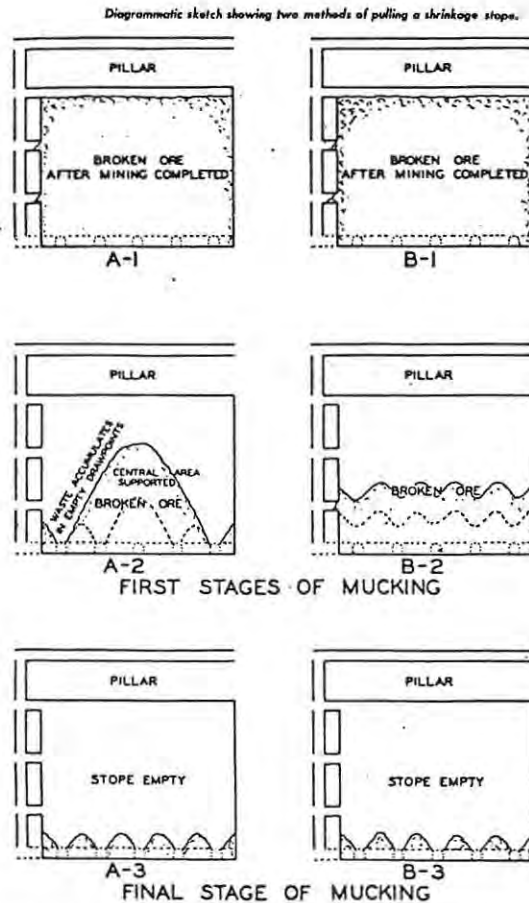
FIG.9.8 (Hopper & Moore, 1957)

9.4 Economics of shrinkage

Pugh and Rasmussen (1982), working under contract for the US Bureau of mines, studied the costs and productivities of three different shrinkage stoping layouts, shown in Fig. 9.9, 9.10 and 9.11, for different orebody widths and ground conditions. A summary of the development costs is shown in Fig. 9.12, while Fig. 9.13 summarizes the overall costs and productivities. The costs are in 1977 American dollars and the constraints are specified in the figures and tables. Productivities vary from 16.2 to 24.8 tons per miner shift, and costs from $4.57 to $7.25 per ton.
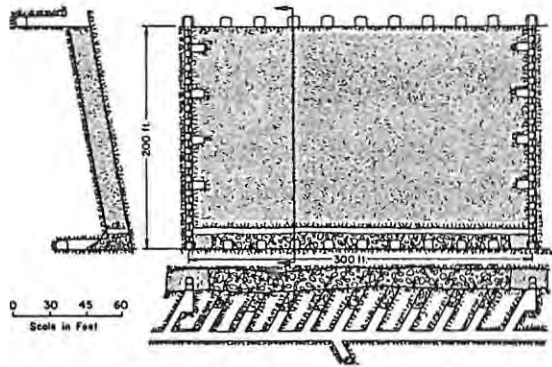
Mechanized shrinkage stoping with drawpoints developed on the haulage level by undercutting. Metric equivalent: ft × 0.3048 = m.

FIG.9.9 (Pugh & Rasmussen,1982)
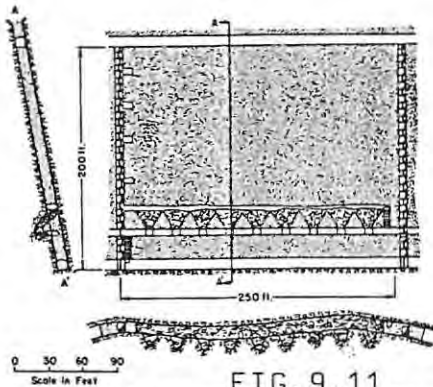
FIG.9.11
Mechanized shrinkage stoping with an intermediate scram drift and belled drawpoints. This differs from both Figs. 1 and 2 in that either the track or trackless heading is driven into the vein. Metric equivalent: ft × 0.3048 = m.

(Pugh & Rasmussen,1982)
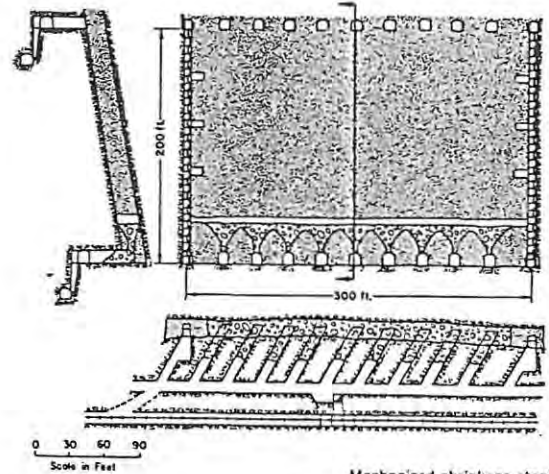
Mechanized shrinkage stoping with an intermediate scram drift and belled drawpoints. Metric equivalent: ft × 0.3048 = m.

FIG.9.10
(Pugh & Rasmussen, 1982)

Shrinkage Stoping Development Cost Summary*

Example	18-ft Wide Vein Mechanized Shrink Stope Trackless Development of Haulage With Crosscuts to Vein and Undercutting Stoping Blocks	18-ft Wide Vein Mechanized Shrink Stope Rail Haulage With Ramp to Scram Drift and Development of Stope With Belled Drawpoints	10-st Wide Vein Conventional Shrink Stope With Scram Drift Development in Ore Block and Belled Drawpoints
Haulage level development	10 x 10-ft bald trackless $71.36 per ft × 300 ft = $21,408	9 x 8-ft bald track drift $98.25 per ft × 300 ft = $29,475	10 x 10-ft timbered drift $139.88 per ft × 250 ft = $34,970
Scram drift development		10 x 10-ft bald sublevel $68.45 per ft × 300 ft = $20,535	7 x 7-ft bald intermediate $71.52 per ft × 246 ft = $17,594
Crosscuts to vein	10 x 10-ft bald, 10 each x 30 ft $68.45 per ft × 300 ft = $20,535	10 x 10-ft bald, 10 each x 30 ft $68.45 per ft × 300 ft = $20,535	7 x 7 ft bald, 9 each x 15 ft $72.12 per ft × 135 ft = $9,738
Undercut vein	10 x 18-ft bald sublevel $73.75 per ft × 300 ft = $22,125		
Drawpoint development		L.S. 9 each, $3,109 × 9 = $27,981	L.S. 9 each, $1,996 × 9 = $17,964
Raise development	7 x 7 ft Alimak $82.47 per ft × 191 ft = $15,752	7 x 7 ft Alimak $82.47 per ft × 191 ft = $15,752	7 x 7 ft Alimak $82.47 per ft × 191 ft = $15,752
Orepass and transfer chute development	5-ft diam orepass (shared) $55.10 per ft × 200 ft × ½ = $3,673	7 x 7-ft raise to sublevel (shared) $82.47 per ft × 30 ft × ½ = $825	7 x 7-ft raise to sublevel $82.47 per ft × 30 ft = $2,474
Chute fronts		L.S. (shared), $3,200 × ½ = $1,067	L.S., $3,200 × 1 each = $3,200
Cutout from manways for stope access	10 ft x 10 each $53.21 per ft × 100 ft = $5,321	10 ft x 8 each $53.21 per ft × 80 ft = $4,257	10 ft x 10 each $53.21 per ft × 100 ft = $5,321
Cribbed manway and ramp to sublevel		10 x 10-ft ramp (shared) $71.36 per ft × 150 ft × ½ = $3,568	6 x 6 ft cribbed $78.25 per ft × 30 ft = $2,348
Total cost development	$88,814	$123,995	$109,089
Total ore developed	300 × 200 × 18 ft × $\frac{162}{2000}$ = $87,480	200 × 300 × 18 ft × $\frac{162}{2000}$ = $87,840	200 × 250 × 10 ft × $\frac{162}{2000}$ = 40,500
Total cost per st of ore developed	= $1.02	= $1.42	= $2.69

*Metric equivalents: ft × 0.3048 = m; st × 0.907 184 7 = t.

FIG.9.12 (Pugh & Rasmussen,1982)

Shrinkage Stopes—Costs and Productivities

		Vein Width 10 ft		Vein Width 18 ft				
Unit Operations	Unit Cost $/st of Ore	Narrow Vein Shrink Stopes with Belled Drawpoints, Good Ground Conditions		Mechanized Shrink Stopes with Vein Undercut, Good Ground Conditions		Mechanized Shrink Stopes with Belled Drawpoints, Good Ground Conditions		Qualifications
		A	B	C	D	E	F	
Development								1. 8-hr shift, 6.5 hr effective time at face.
18-ft wide vein (Fig. 1)	$1.02	—	—	1.02	1.02	—	—	2. Labor rate $10.50/hr, includes bonus and fringe.
18-ft wide vein (Fig. 2)	$1.42	—	—	—	—	1.43	1.43	
10-ft wide vein (Fig. 3)	$2.69	2.69	2.69	—	—	—	—	3. Unit operation costs include labor.
Drill and Blast								4. Equipment costs include maintenance parts and labor, tires, and fuel. Equipment depreciation is not included.
Breasting—10-ft vein, 51.8 st jackleg	$1.58	—	1.58	—	—	—	—	
Breasting—18-ft vein, 93.2 st jackleg	$1.38	—	—	1.38	—	1.38	—	
Back stope—10-ft vein, 35.2 st stoper	$1.51	1.51	—	—	—	—	—	
Back stope—18-ft vein, 63.4 st stoper	$1.24	—	—	—	1.24	—	1.24	
Mucking								5. All materials are based on 1976 prices.
Slusher—25 hp*	$1.10	1.10	—	—	—	—	—	6. Machine ownership costs excluded.
LHD—1 cu yd diesel	$0.92	—	—	—	0.92	—	—	
LHD—2 cu yd diesel	$0.70	—	—	0.70	—	—	0.70	
LHD—5 cu yd diesel	$0.64	—	—	—	—	0.64	—	
Cavo 511 pneumatic	$0.78	—	0.78	—	—	—	—	
Ground Support								
Rockbolts—10-ft wide vein	$1.52	1.52	1.52	—	—	—	—	
Rockbolts—18-ft wide vein	$1.47	—	—	1.47	1.47	1.47	1.47	
Stope Preparation								
Cribbed manway—10-ft vein width	0.43	0.43	0.43	—	—	—	—	
Total Cost per Short Ton		$7.25	$7.00	$4.57	$4.65	$4.91	$4.83	
Stope Crew Size		3	3	3	3	3	3	
Productivity (st per Minershift)		16.6	16.2	22.2	17.1	24.8	18.8	

*Metric equivalents: ft × 0.3048 = m; cu yd × 0.764 554 9 = m³; hp × 745 699 9 = W; st × 0.907 184 7 = t.

FIG.9.13 (Pugh & Rasmussen,1982)

10. CUT-AND-FILL

10.1. The application of cut-and-fill

The cut-and-fill method may be adapted to almost any type of ore-body with a relatively high vertical extent, but is probably best employed where the ore has poor continuity and where most types of bulk mining would produce excessive dilution. Probably, the only characteristic demanded is that the ore must be strong enough to support itself over the width of the stope, or to be supported economically by means of roofbolts or cable bolts. The size and shape of the stope may be readily changed to follow almost any shape of orebodies, and good planning, systematic sampling, and careful supervision will produce high ore recoveries and a product with less dilution than any other stoping method.

The advantages of cut-and-fill are: the continuous and extensive sampling of the orebody as it is being mined; minimum development before mining is started; selective mining can be used to reduce dilution; its versatility makes it possible to efficiently mine plunging orebodies; the openings are small and there is less dilution from slabbing of walls than in other methods; a change to another method can be readily made; equipment investment is relatively small; the ore is removed as fast as it is broken so that capital is not tied up and there are no fire hazards or oxidation problems; the fill supports weak walls and also maintains the general stability of the mine (there are no stopes to collapse suddenly); the reduced subsidence can enable mining to take place below bodies of water; orebodies can be mined at great depths; and the returning of the bulk of the mill tailings underground avoids surface disposal problems.

The disadvantages of cut-and-fill are: the ore production is cyclical; the method can be labour-intensive and requires skilled miners; the productivity is moderate; the cost of fill can be high; the disposal of tailings slimes without coarse sand can give problems of tailings dam stability and tailings pond restoration; the personnel must work under freshly blasted ground which creates safety problems, although the exposed areas have not time to deteriorate; proper ventilation is difficult and expensive; and costs are higher and a high grade orebody is required.

10.2 Layout planning

In cut-and-fill mining, flat or inclined slices or ore on the backs
of the stopes are blasted down in successive lifts. The broken ore is
removed and the stope is filled with waste, leaving just enough room
between the hangingwall and the waste pile or sand fill to provide working
space. This cycle of operations is successively repeated, and mining
retreats from the bottom of the stope in an upwards direction. Orebodies
are stoped from one level to the next keeping a relatively horizontal
working face (except for some forms of traditional cut-and-fill with
scraper-mucking), and the fill prevents the large scale collapse of the
country rock.

An important factor in the planning of cut-and-fill systems is the
availability of an economic backfill material and of an efficient trans-
port system for this material. The fill may consist of waste sorted
from the ore in the stope, waste rock from development work or from
waste stopes excavated to provide filling material, or (especially in
larger operations) of sand, gravel, and deslimed (sizes less than 10 to
20 microns removed) mill tailings pumped into the stopes. To the
"hydraulic fill" (deslimed tailings), cement is normally added in concen-
trations ranging from about 8% to 20%.

In tabular steeply-dipping orebodies of narrow to moderate width,
the ore is usually mined in longitudinal stopes. The stopes may extend
to the full length of the oreshoots or may be limited in length by pillars,
which will reduce the exposed span of the openings. Stopes up to 20 m
in width may be mined under competent hangingwalls. In wider orebodies
transverse pillars are left, and the layouts normally consist of a series
of intercommunicated transverse stopes.

In order to maximize the ore recovery, as few pillars as possible
should be left behind; if possible these pillars should be left in waste
rock (waste pockets within the orebody can be blasted and the broken
material used as stope fill as well). Mining should commence on alter-
nate levels or even further apart to eliminate sill pillars; however,
especially in short along the strike but long along the dip orebodies,
increasing the stope height could reduce the overall mining rate.

Other general rules for laying out longitudinal stopes include the following ones: (i) whenever possible start mining at the lowest elevation; (ii) follow a retreat sequence that would result in a lower maintenance of haulages cost and in the right mining sequence according to a definite grade control criteria (e.g. high-grading during the early life of the mine and producing at a constant grade and tonnage throughout the rest of the productive life); and (iii) schedule an early extraction of pillars to avoid excessive deterioration.

The practice of controlling the overall output of a mine by temporarily stopping the mining of certain stopes and starting again the interrupted mining of other stopes is bad. To obtain a high productivity, stopes once started should proceed without interruption, and the only tool to control the overall output would be the alteration of the stoping rates. However, modern high productivity stoping aims to achieve and maintain the maximum possible output from a stope. In this case, modifying the rate of stoping cannot be used to control the grade and tonnage of the overall output of the mine without reducing the productivity, and the only way to control them is through a carefully done scheduling of the development and mining of the different stopes (which requires a good knowledge of the orebodies prior to their development).

Consider an orebody as in Fig. 10.1 (McMillan and Ferguson, 1982): 274.3 m (900 ft) high, 228.6 m (750 ft) long on strike, and averaging 4.6 m (15 ft) in width; it represents approximately 900 000 tons. The orebody could be mined advancing one long single face through all the levels from bottom to top (see Fig. 10.1.a), and the length of the orebody could accommodate five 45.7 m long stopes; each of these stopes would produce approximately 1800 tons per month, and the mine would have a life of 10 years.

The staggered pattern shown in Fig. 10.1.b would reduce the problems of hangingwall failure. To reduce the maintenance of accesses, retreating towards the shaft constitute a good practice: stopes farthest from the shaft are mined first and on completion haulage drifts below can be abandoned. Such a sequence, shown in Fig. 10.1.c, will take longer than (b) to reach full production (at 1.5 years per level advance, an extra
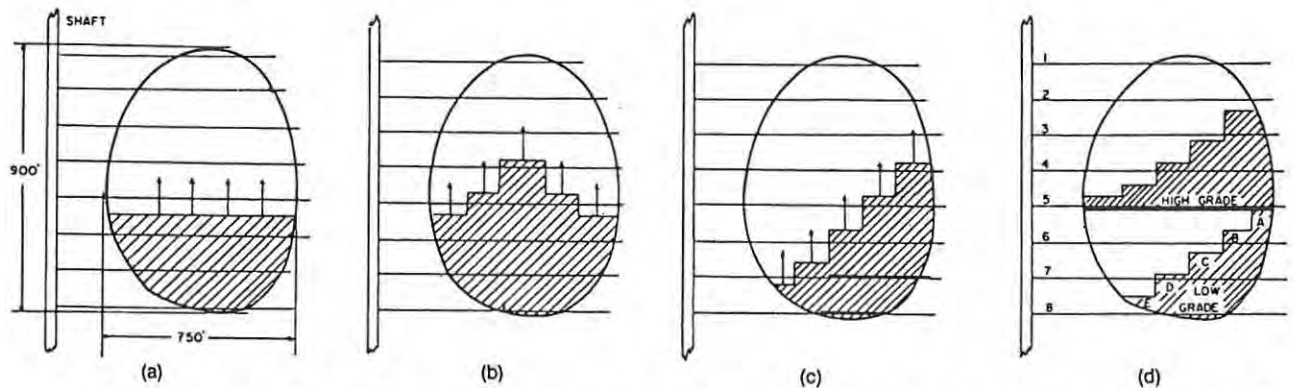
Stope planning for longitudinal flat-back cut-and-fill. The ore zone is 274 m (900 ft) high, 228 m (750 ft) on strike, by 4.5 m (15 ft) wide, representing 907 000 t (1,000,000 st).
(a) Advance one long single face through all levels from the bottom to the top in five stopes, each 45 m (150 ft) long. (b) Hanging wall failure would be reduced by staggering individual stope faces. (c) A sequence of retreating toward the shaft reduces travelway maintenance but takes longer to achieve full production. (d) Mining from several horizons creates problems of sill pillars.

FIG.10.1   (McMillan & Ferguson,1982)

three years are required to reach full production of 9000 tons per month), the initial rate of production could be too low, and if the grade of the orebody decreases towards its edges it may cause too great a fluctuation in grade over the orebody life or too low an initial grade.

To overcome these problems mining can commence from more than one elevation, as shown in Fig. 10.1.d. This solution, however, creates a further problem: a sill pillar is left immediately below the second horizon (i.e. below level 5 in Fig. 10.1.d). This relatively high grade pillar will have to be removed by a modified mining method, such as square-set stoping or undercut-and-fill, because of the crushed ground, and mined in portions as soon as the stopes (A, B, C, D and E) successively reach the sill level.

The previous example shows some of the parameters and criteria involved in the adoption of an overall mining sequence. This decision is intimately related to the principles involved in stope design, and it is normally made based on a trial-and-error process.

Due to the flexibility of the method and the variability of the ore zones, the cut-and-fill layout is usually designed on a stope basis. In discontinuous orebodies the size of the stope is normally determined by the ore boundary, while in continuous orebodies the length of the

stope may be determined by the length of time that each of the cycles (preparation, backfill, drilling, blasting and ore extraction) requires. The size and continuity of the orebody also determine the mucking method and equipment (scrapers or LHD units) to be used, which in turn will define the developments required and the size of the openings.

The stope height is controlled by the necessity of optimizing the overall mining schedule along the lines mentioned in the previous example. However, in conventional cut-and-fill the level interval is usually maintained at 45 m to 60 m (150 ft to 200 ft), since the miners have to enter and leave the stope each shift. The stope height usually comprises two, or even more, access levels.

The height of the mining section (opening between the fill and the back of the stope) is usually determined by the strength of the wall rock and the amount of roofbolting required. Fig. 10.2 shows some rock-mechanics problems in cut-and-fill mining. When mining closely adjacent stopes, one should be kept in advance of the other to separate the two zones of peak abutment stress that develop above the stopes. The stope backs should be orientated parallel to the trend of the major joints to give a smooth back, easier to roofbolt and without wedges of rock that can drop out. The back should be arched, if necessary, removing the ore that tends to sag causing the joints to open and blocks to drop out (the height of the arch is about 0.1 to 0.2 of the stope width).

Once the height of the mining section has been decided, the appropriate drilling method and equipment can be chosen. In Fig. 10.3 it can be seen that drilling horizontal blastholes the size of the opening is considerably reduced and less damage is caused to the stope backs. Nevertheless, with vertical or inclined blastholes large sections of the roof can be drilled without interruption and large rounds can be blasted (the drilling equipment is simpler as well).

The number and location of ore passes are controlled by the rate of production and by the decrease in efficiency of the mucking equipment for long runs (30 to 40 m for wheelbarrows and scrapers, 40 m for Cavo-type loaders, and up to 120 m for LHD loaders are roughly the maximum efficient run distances). Most ore pass systems are designed to accommodate
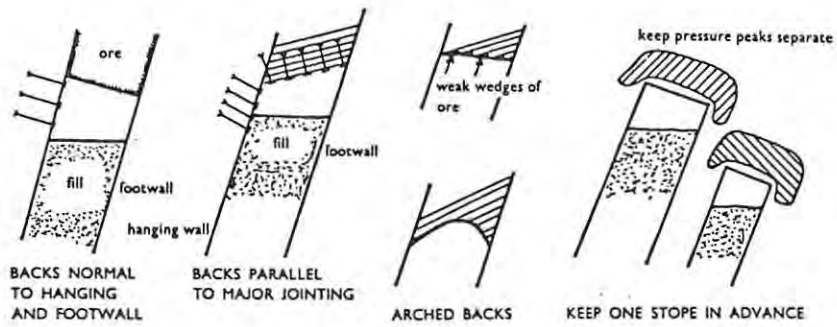
Working bad ground

FIG.10.2 (Thomas,1978)
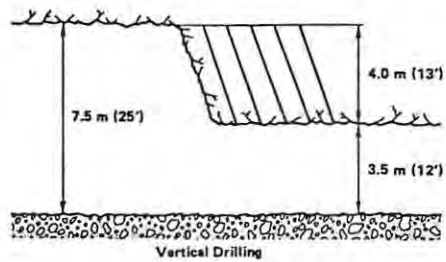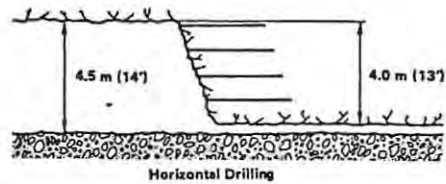
Comparison of cut-and-fill drilling systems.

FIG.10.3 (Hamrin,1982)

direction changes should the stope size change. They may be lined with timber, steel, or concrete, and the wear factor should be assessed to estimate the tonnage that will be drawn through the opening.

The evolution of the cut-and-fill mining used by the Boliden group in 14 of its copper, lead and zinc mines in Sweden (World Mining, October 1982, pp. 64-67), reflects the changes in mining equipment and, consequently, in layouts and mining techniques experienced by cut-and-fill mining throughout the world. During the 1960's, the traditional mucking with scrapers was replaced by air powered auto loaders (Cavo

type) mucking, pneumatic drill jumbos were introduced replacing the
jackhammers with air legs, and productivity increased by roughly 20%
(see Figs. 10.4, 10.5 and 10.6).   During the 1970's diesel powered
wheel loaders (LHD units) replaced the air powered auto loaders,
hydraulic drill jumbos were introduced, and productivity increased by
nearly 100%.   The layouts were modified accordingly and a short review
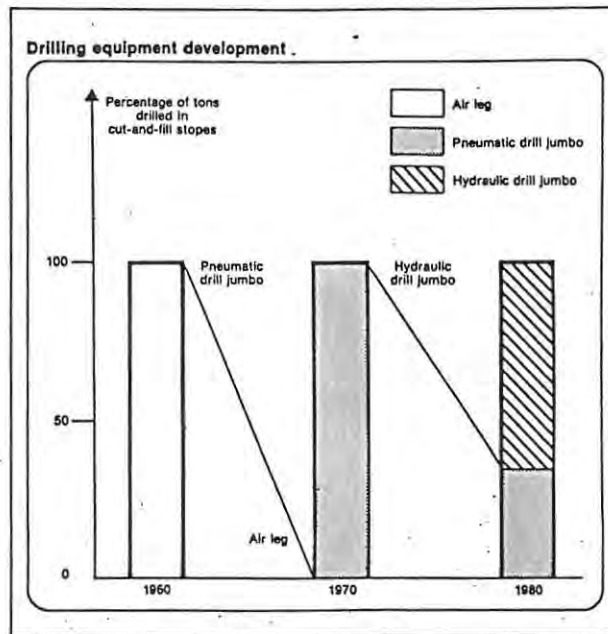of the main layouts is presented in the following pages.

FIG.10.4 (World Min.,Oct.,1982)
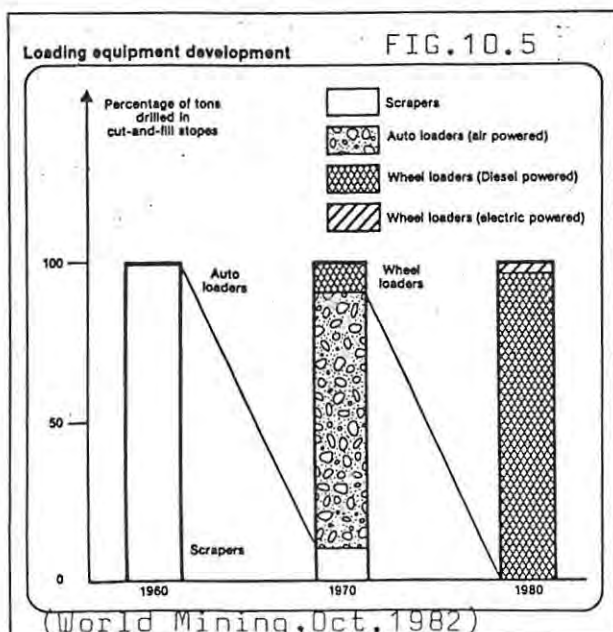
FIG.10.5

FIG.10.6
(World Mining,
Oct.,1982)

The "conventional" or first generation cut-and-fill used until the 1960's was characterized by vertical hand-held drilling (air legs), scraper mucking (wheelbarrows and handlashing in early versions), filling with waste and/or gravel, and timbered ore passes and travelling ways in the fill. Fig. 10.7 shows a typical flatback cut-and-fill stope (the Malartic gold mines, in Quebec), in which the cuts or flat slices are 8 ft (2.4 m) in height and the backfill is placed to within 7 ft (2.1 m) of the back. At the Hollinger gold mines (Canada), the use of 36 in. wide Calumet-type scrapers, pulled by 25 or 30 hp DC motors and using 1/2 in. to 7/8 in. ropes, resulted in an ideal mining plan consisting of ore passes about 150 ft (45.7 m) apart with a backfilling raise midway between them. However, the ideal plan could rarely be followed because of the irregularities of the orebody outlines; for example, in the diagram in Fig. 10.8 a third set of ore pass and travelling way has been added between two sets spaced according to the plan, in order to mine the vein branching away from the main stope.

The sill pillars above the stopes were recovered in the way shown in Figs. 10.9 (for gravel fill stopes) and 10.10 (for hydraulic fill stopes).

Alternatively, the "conventional" cut-and-fill stopes were mined using inclined layouts, whose three varieties are shown in Figs. 10.11, 10.12 and 10.13. At the Malartic gold mines (Dempsey and Kennedy, 1957), in Canada, the ideal maximum distance of scraping of 36.6 m in flat stopes was reduced to 22.9 m in inclined layouts, because in most cases the operator could not see the scraper and because it was necessary to carry equipment to and from the working face.
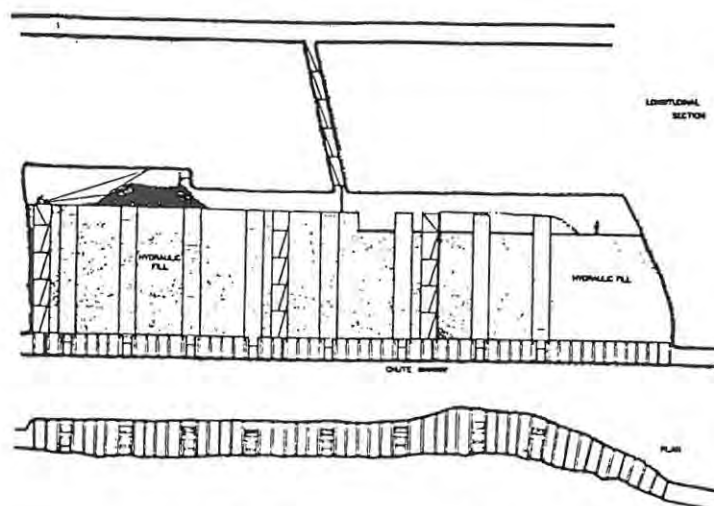
Plan and section of flatback cut-and-fill stope.

FIG.10.7 (Dempsey & Kennedy,1957)
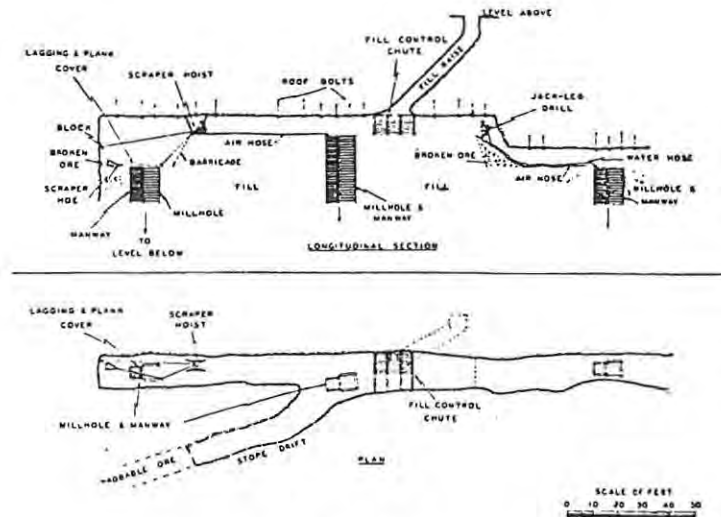
Plan and section showing cut-and-fill.

FIG.10.8 (The Staff,1957)
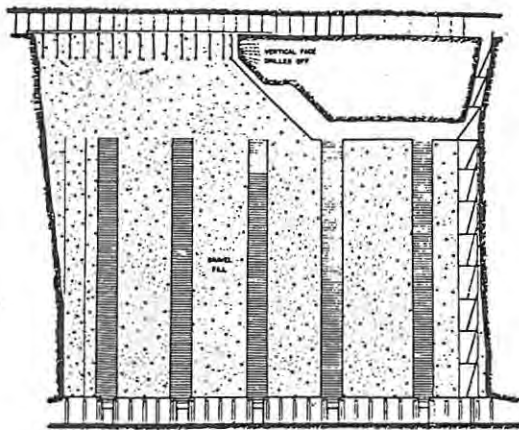
Section showing method of sill recovery in gravel fill areas.

FIG.10.9 (Dempsey &
Kennedy,1957)
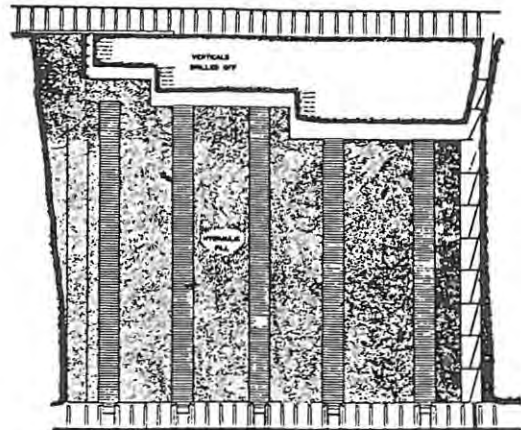
Section showing experimental method of sill recovery in hydraulic fill areas.

FIG.10.10 (Dempsey &
Kennedy,1957)

The table in Fig. 10.14 shows the costs (in 1957 Canadian dollars) at Malartic, and it is noticeable that inclined cut-and-fill is more costly than the flatback layout, the increase in scraper efficiency being overcome by the increase in costs of break-labour, muck-labour and timbering (labour and supplies). The productivity of this conventional cut-and-fill is about 9 to 12 tons per underground manshift (the labour working in these stopes was normally not specialized; they were all-round miners, because of the long and difficult access to the stopes).

Layout of inclined cut-and-fill steps for slusher mucking.

FIG.10.11
(Dempsey &
Kennedy,1957)

Double inclined cut-and-fill stopes two chutes.

FIG.10.12 (Dempsey &
Kennedy,1957)

Double inclined cut-and-fill; single chute.

FIG.10.13 (Dempsey &
Kennedy,1957)
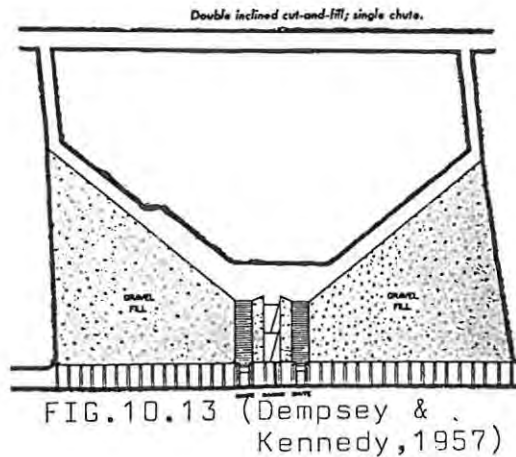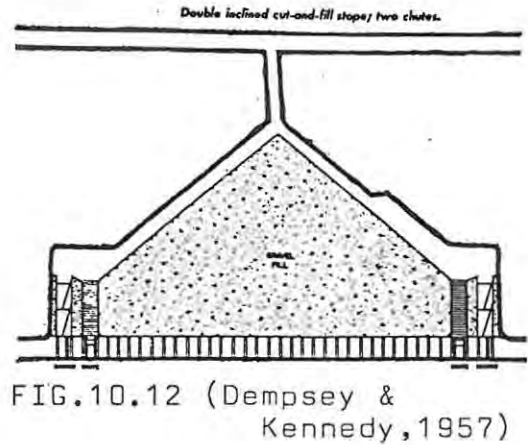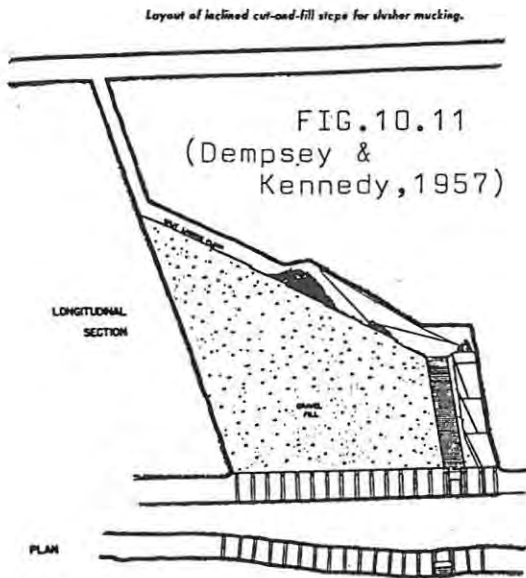
Type	Break Labour	Expl.	Muck Labour	Tbr. Labour	Tbr. Supply	Fill Labour	Fill Supply	Bonus	Total
Incline	.291	.198	.343	.182	.301	.066	.236	.272	1.889
Sill Recovery	.235	.154	.256	.177	.184	.064	.387	.275	1.732
Flatback (Sand Fill)	.229	.204	.263	.118	.398	.103	.195	.212	1.722
Flatback (Hydro Fill)	.208	.176	.295	.135	.292	.083	.093	.239	1.521

FIG.10.14 (Dempsey & Kennedy,1957)

The conventional cut-and-fill stoping is still a viable alternative for veins that are too narrow for rubber-tyred equipment (i.e. stoping widths of 1.2 to 3.0 m). The grade of the ore must be high and the method must be quite selective to be commercial.

In the early forms of mechanized cut-and-fill (1960's) the LHD units and the drill jumbos were captive in the stope; they had to be serviced and maintained in the working stope and the equipment capacity had to be closely matched to the stope production requirements. At this stage, horizontal drilling with pneumatic jumbos, hydraulic filling with deslimed tailings, steel-lined ore passes and manways placed in the fill, and Cavo-type pneumatic autoloaders, and LHD units later, were introduced.

The layout used at Mount Isa, shown in Fig. 10.15 (Thomas, 1978), is an example of these early stopes. A 12 to 15 m high sill was left above the haulage level and the initial access to the stope was through a 1 in 8 sill incline; this access was sealed as mining progressed. During more advanced stages, the personnel, materials and fill were introduced into the stope through raises in the back. Steel-lined ore passes were driven along the footwall contact of the stope, their spacing controlled by the capacity of the loading equipment, the mining rate and the irregularities of the orebody outlines (generally, about 40 m spacing between ore passes); in some mines the ore passes were developed in the footwall country rock, about 8 to 10 m from the ore contact for maximum stability.
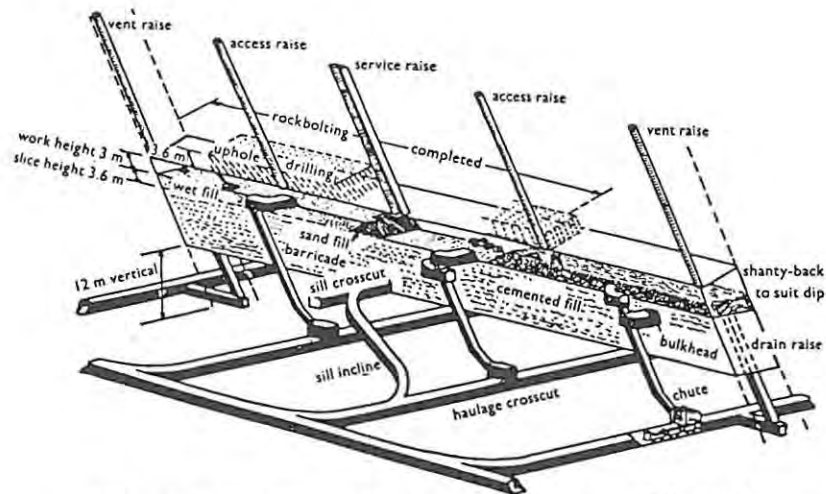
Diagrammatic layout of a MICAF stope
(original courtesy of Mt Isa Mines)

FIG.10.15
(Thomas,1978)

The Cobar copper mines in New South Wales were among the first operations to use access inclines (in the late 1960's). These orebodies dip at 70° to 80° and their widths range from 4.5 m to 21 m, the average being 10.7 m. The stopes were opened up over the full strike length of the orebodies, ranging from 150 m to 365 m in length. Main levels, with servicing and workshop facilities, were established at vertical intervals of 180 m. To have access to the stopes, a 1 in 7 incline system was developed in the footwall country rocks, about 15 m away from the ore contact. Since every lift, or slice of roof ore mined, is 4.5 m thick, 4 m x 3 m crosscuts were driven from the incline to the stope at 4.5 m vertical intervals (see Fig. 10.16). The ore from the stope is trammed directly to the main ore passes when the stope access is close to a transport crosscut, otherwise the ore is tipped into a secondary ore pass off the incline system with a drawpoint located off a lower transport crosscut to minimize the tramming distances (see Fig. 10.17).
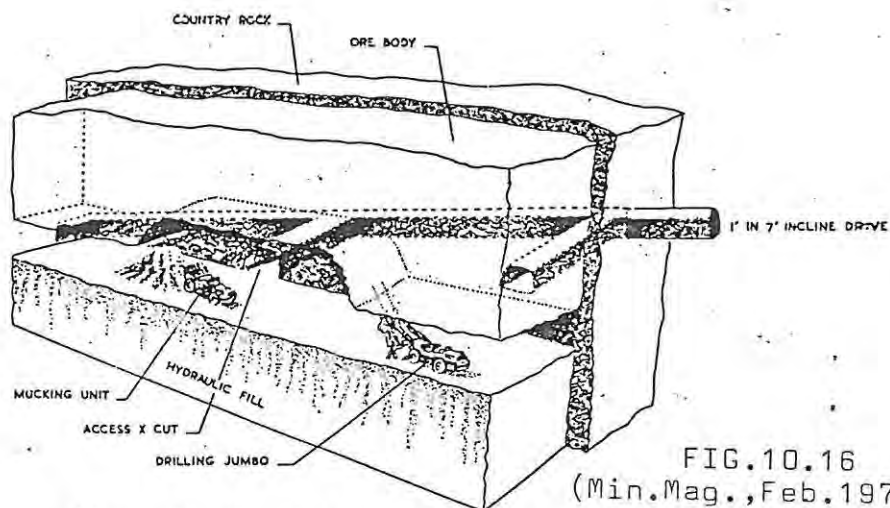
FIG.10.16
(Min.Mag.,Feb.1972)

Idealised section of stoping method showing angled drilling, loading, followed by hydraulic fill

General layout of ore flow from stopes to shaft. ....Copper ore flow - - - Copper-Zinc ore flow——Ore flow common to both ores. 1. Copper ore. 2. Copper zinc ore. 3. Stope ore pass and draw points. 4. Transport cross-cut. 5. Copper ore pass. 6. Copper-zinc ore pass. 7. Crusher station. 8. Inclined conveyor. 9. Copper fine ore storage. 10. Copper-zinc fine ore storage. 11. Loading pocket conveyor. 12. No. 2 shaft hoisting.

FIG.10.17
(Min.Mag.,Feb.1972)

In 1977 G.M. Pugh and D.G. Rasmussen (in: Pugh and Rasmussen, 1982), working under contract with the US Bureau of Mines, studied new forms of mechanization for vein mining and they developed the cut-and-fill system shown in Fig. 10.18.
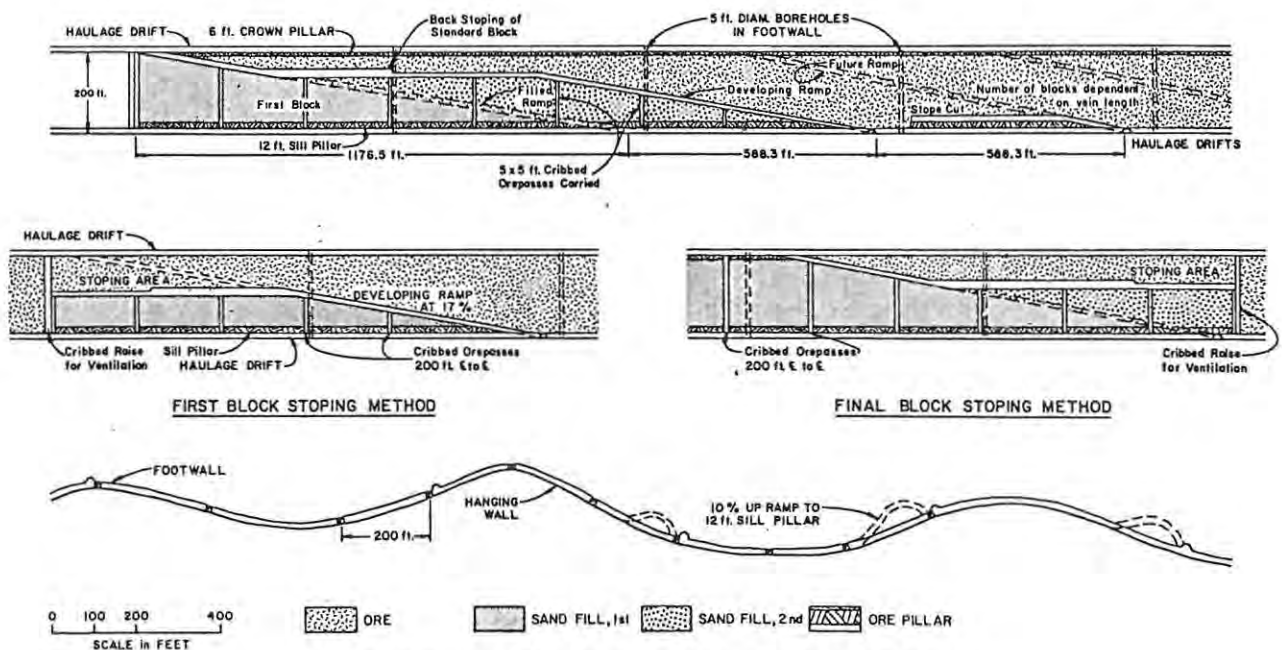
Highly mechanized cut-and-fill stoping. Metric equivalent: ft × 0.0348 = m.

FIG.10.18 (Pugh & Rasmussen,1982)

The ramp access is developed at 17% on top of the fill as the
stope advances, thus dividing the stope into two blocks.   The first
block, located under the ramp, is mined as the ramp advances.   When
the mining of the first block has been completed and the ramp has
"holed" into the upper level, the second block is mined retreating
towards the upper level losing the ramp access from the bottom one
(raises in both ends of the stope are required for ventilation).
Wooden or steel-lined ore passes are developed in the fill.   Figs. 10.18,
10.19 and 10.20 show the dimensions of the different elements of this
proposed system and details of some operations.   The overall mining
sequence is the one shown in Fig. 10.21.   This highly mechanized
system will show significant improvements over conventional practices
in productivity, operating costs, and safety;   the costs and productivity
of this system are considered in the section on economics of cut-and-fill.
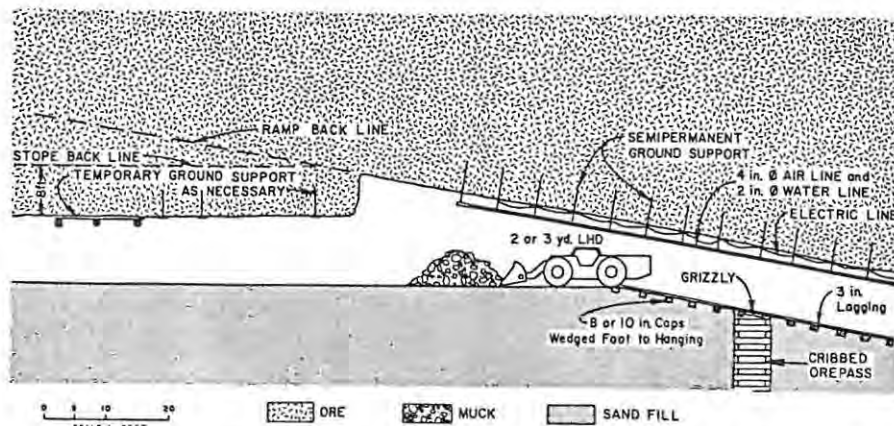
First block unit operation. Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m; yd × 0.914 4 = m.

FIG.10.19   (Pugh & Rasmussen,1982)
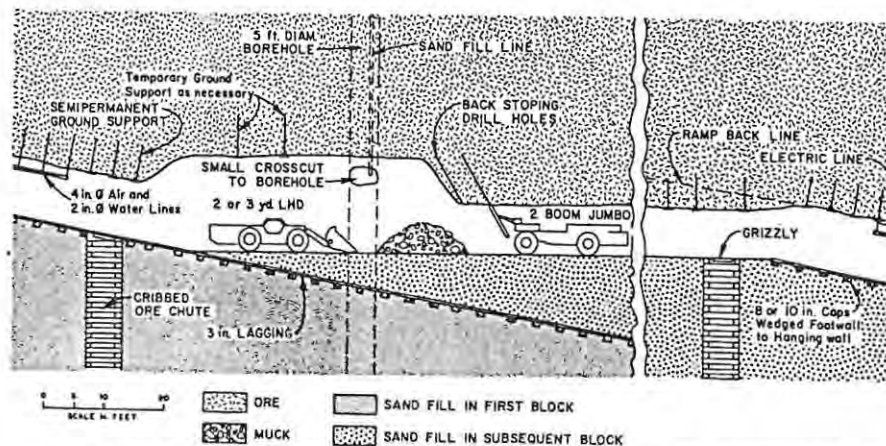
Back stoping operations. Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m; yd × 0.914 4 = m.

FIG.10.20   (Pugh & Rasmussen,1962)

Variation of mechanized cut-and-fill for a short vein strike length. Metric equivalent: ft × 0.3048 = m.
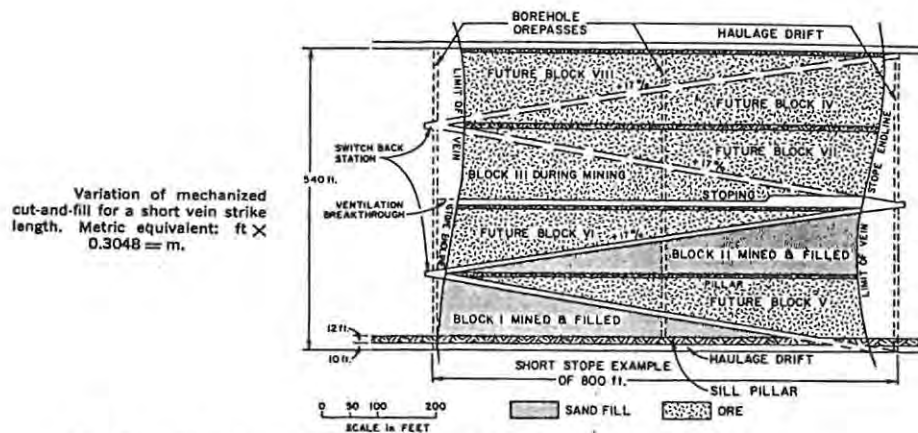
FIG.10.21   (Pugh & Rasmussen,1982)

It was estimated that the new ramp-in-stope form of cut-and fill would have a development cost 28.6% higher than that of conventional mechanized cut-and-fill, and a 50% higher equipment cost, but it would reduce the labour needs by 35% and the operating costs by 30%.   This new system requires more regular orebodies than it is required for conventional methods.

When the orebodies exceed about 10 m in width stoping by simple longitudinal cut-and-fill methods becomes difficult, and often impossible for widths greater than 20 m.   To overcome the rock mechanics problems, transverse stopes with rib pillars are mined.   Fig. 10.22 shows the transverse stoping used at the Strathcona mine and the mining sequence followed there; the extraction of the rib pillars is by undercut-and-fill, which is most efficient when the pillar width is less than 6.1 m (20 ft).

At the New Broken Hill zinc-lead mine, the approximately 28 m wide steeply-dipping orebodies are divided in a series of E-shaped panels, as shown in Fig. 10.23.   An access and ventilation raise in ore connects the working level with the haulage level above.   All other connections to the stope (travelling ways and ore passes) are from below through timber-lined passages in the fill.   All the raises are on the side of the pillars to facilitate their final recovery in a second stage of cut-and-fill mining.
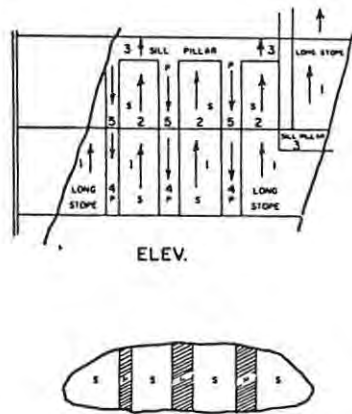
FIG.10.22 PLAN

Stope and pillar layout at Strathcona
mine for a cut-and-fill mining block between
the 1700 and 2700 levels. The numbers
represent the sequence of mining.

(McMillan & Ferguson,1982)

AR = access raise
L/T = ladder/timberway
OP = ore pass
SB = sand barricade
1-4 = mining sequence
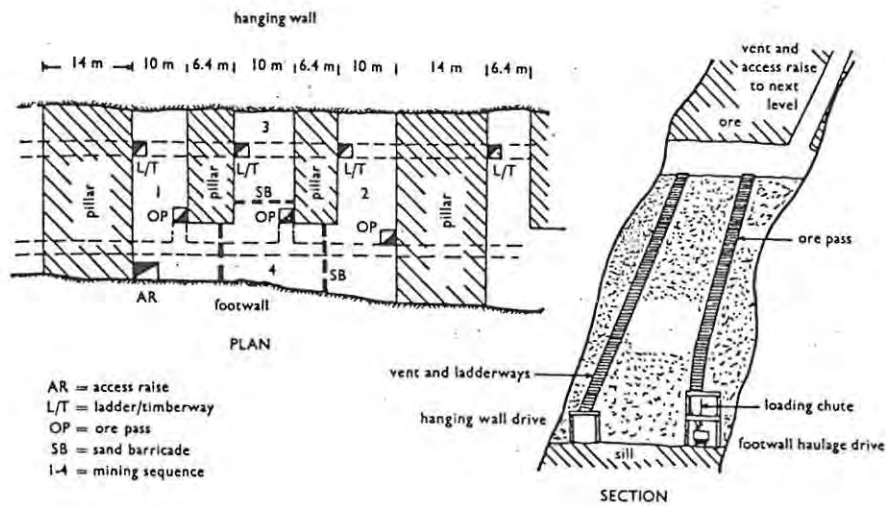
Cut and fill stopes at New Broken Hill Consolidated
FIG.10.23 (Thomas,1978)

10.3. Evaluation and grade control

Cut-and-fill is applicable under a wide range of conditions : from
small to large deposits of irregular outline, and from flat-dipping deposits
to dips of 90°, although filling operations are easier in steep deposits.
The minimum mining width is roughly 1.2 m, and irregular ore lenses can
be grouped in interconnected stopes, as shown in Figs. 10.24 and 10.25.
The second example is from the Renström massive zinc-copper-lead-silver
sulphide deposit in Sweden. The fill stabilizes the stopes and makes
possible the mining of stopes very close to each other.
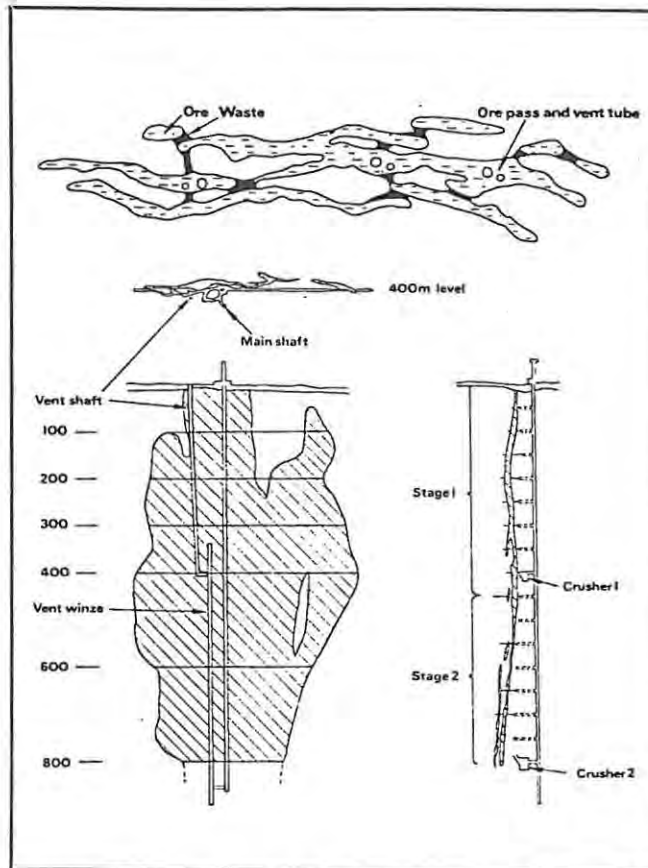
Above: Diagrammatic plan of orebody at 780m level,
showing ore pass and vent tube arrangements
Below: Section of orebody down to 800m level
FIG.10.25
Mining Magazine—September 1971, Vol. 125, No. 3
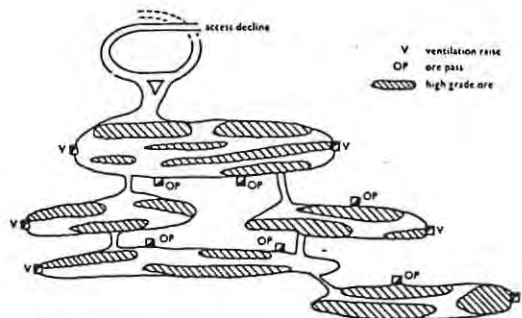
Plan of interconnected cut and fill stopes
FIG.10.24 (Thomas,1978)

Stringers of valuable ore running out into the country rock can
easily be mined by cut-and-fill. In some situations the mining of these
may involve splitting the stope in two, and the access and mucking
problems must be considered. In the high-productivity forms of cut-and
fill the mining of these off-shoots could destroy the mining schedule,

reduce the overall productivity and, therefore, be uneconomic; this problem is accentuated by the fact that cut-and-fill is a cyclic method.

In some cases high grade veins narrower than a practicable stoping width may be taken by "resuing", so as not to dilute that ore with waste. In resuing, one wall is shot down first and left in the stope for fill, after which the vein is stripped.

In cut-and-fill mining, the pattern of diamond drilling and the cross-sectional methods of evaluation of ore reserves are similar to those used to evaluate orebodies mined by sublevel stoping. Nevertheless, the greater selectivity and lower dilution of cut-and-fill change the results, and ore reserves recoverable by cut-and-fill frequently differ from those recoverable by sublevel stoping from the same orebody, as shown in Fig. 10.26 for a very simple situation. Ore recoveries of 90% to 100% and dilutions of less than 10% are common in cut-and-fill mining.

Sublevel Stoping

Cut-and-Fill Stoping
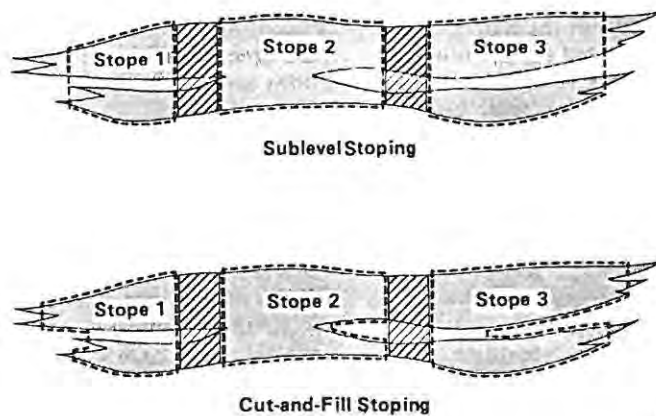
FIG.10.26
(Hamrin,1982)

Stope-outline variations for different mining methods.

The selectivity of cut-and-fill mining very often makes difficult, or even impossible, an economic accurate evaluation of recoverable ore reserves. The method permits to mine selectively small off-shoots, and at the same time to leave behind small pockets of waste (as pillars or as stope fill), and in an irregular orebody the delineation of all these small irregularities would require a very close grid of diamond drilling, which would be extremely expensive. The answer to this problem is to

use "reconciliations" of ore reserve estimates, muck grab samples, drawpoint sampling and skip sampling with mill headgrades (as discussed in the chapter on sublevel stoping) to estimate empirical factors to be applied to the ore reserve estimates; these factors would account for the changes, in the estimates, due to these small irregularities in the orebodies.

The presence of these small scale irregularities in the outline of the orebodies and in the distribution of the ore within them, makes secondary sampling necessary. During the stoping stage, sludge sampling is used to delineate the outline of the orebodies (and the distribution of the values) in the next slice to be blasted or in the one being blasted. Fig. 10.27 shows a pattern of percussion drilling used to delineate the orebody in the next slice. In Bolivian tin mines, a frequent sampling practice consists in chip sampling of the back of the stope, and percussion drilling and sludge sampling of the sidewalls, which is done at 3 m spacing along the strike of the ore-bodies; the samples are crushed and panned, and the grade is visually estimated. Whenever it is possible to use them, panning and visual estimates can be advantageous in this process of secondary sampling.
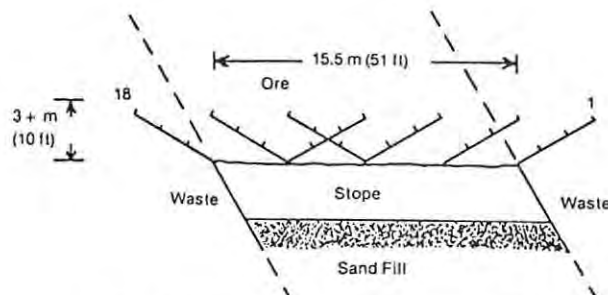
Typical cross section of an open cut-and-fill (OCF) stope showing sample density (51 x 10 x 20 ft = 10,200 cu ft; 10 cu ft/st = 1020 st; 1020 st/14 samples = 73 st/sample).

FIG.10.27 (Tapp,1982)

The dilution by overbreaking is normally very reduced (about 5% or less, depending on the regularity of the orebodies and on the stope width). The accuracy of blasting is normally high in cut-and-fill, and the over-breaking of the sidewalls does not exceed 30 cm; dilution by overbreaking is frequently estimated assuming an overbreaking of 30 cm in each sidewall.

The fill is the other source of dilution. Cement dust has been mixed into the top few centimetres of the hydraulic sand fill with a rotary cultivator to give a hard crust, thus reducing this form of dilution. However, some of the broken ore mixes with the top of the fill and it is very difficult to completely eliminate this dilution. About 5% dilution with fill is an acceptable level.

10.4 Economics of cut-and-fill

Pugh and Rasmussen (1982) analyzed the costs and productivities of four different alternatives of conventional cut-and-fill mining. Their calculations assume: (i) 2.1 m and 5.5 m wide, nearly vertical, vein-like orebodies; (ii) a 200 ft (61 m) vertical distance between haulage levels; and (iii) 300 ft (91.4 m) spacing along strike between sets of ore passes and travelling ways. Other constraints for these four alternatives (A, B, C and D) and the detail of their development costs are given in Fig. 10.28, while Fig. 10.29 summarizes the costs and productivities of these four situations under different constraints. Note that mucking done with scrapers, Cavo and LHD units are alternatively considered. Costs are in 1977 American dollars. The total costs vary from $7.49 to $14.67 per ton, productivities from 7.0 to 30.4 tons per miner-shift, and production rates (for 61 m high and 91.4 m long stopes) from 14 tons to 60.8 tons per shift.

Cut-and-Fill Development Cost Summary

	Case A, 7-ft* Wide Vein Level Developed on Vein Leaving Sill Pillar	Case B, 7-ft Wide Vein Timbered Level Development on Vein, No Sill Pillar	Case C, 7-ft Wide Vein Level Developed in Footwall With Crosscuts in Vein	Case D, 18-ft Wide Vein Level Development in Footwall With Crosscuts to Vein
Haulage level development	9 × 8 ft bald drift $98.25 per ft × 300 ft = $29,475	10 × 10 ft timbered drift, $139.88 per ft × 300 ft = $41,964	9 × 8 ft bald drift $98.28 per ft × 300 ft = $29,475	9 × 8 ft bald drift $98.25 per ft × 300 ft = $29,475
Intermediate development	7 × 7 ft bald drift $71.52 per ft × 282 ft = $20,169	7 × 7 ft bald 1st cut $66.18 per ft × 295 ft = $19,523	7 × 7 ft bald drift $71.52 per ft × 295 ft = $21,098	10 × 18 ft bald drift $74.11 per ft × 300 ft = $22,233
Crosscut to vein			9 × 8 ft bald drift $98.25 per ft × 100 ft = $9825	9 × 8 ft bald drift $98.25 per ft × 150 ft = $14,737
Chute and manway cutouts	4-compartment raise = $7902			
Chute fronts	2 each, $3200 × 2 = $6400	2 each, $3200 × 2 = $6400	2 each, $3200 × 2 = $6400	2 each, $3200 × 2 = $6400
Raise development	7 × 7 ft bald raise (Alimak) $82.47 per ft × 191 ft = $15,752	13 × 7 ft timbered raise $117.25 per ft × 191 ft = $22,395	5-ft diam bored raise $55.10 per ft × 200 ft = $11,020	5-ft diam bored raise $55.10 per ft × 200 ft = $11,020
Total cost development	$79,698	$90,282	$77,818	$83,865
Tons ore developed	$300 × 200 × 7$ ft $\times \frac{162}{2000}$ = 34,020	$300 × 200 × 7$ ft $\times \frac{162}{2000}$ = 34,020	$300 × 200 × 7$ ft $\times \frac{162}{2000}$ = 34,020	$300 × 200 × 18$ ft $\times \frac{162}{2000}$ = 87,480
Cost per st* ore developed	$2.34	$2.65	$2.29	$0.96

*Metric equivalents: ft × 0.3048 = m; st × 0.907 184 7 = t.

FIG.10.28 (Pugh & Rasmussen,1982)

Costs and Productivities for Cut-and-Fill Stoping

Unit Operations	Unit Cost, $/st of Ore	I Conventional Cut-and-Fill With Moderate Ground Conditions	II Back Stope Cut-and-Fill With Moderate Ground Conditions	III Cut-and-Fill Stoping With Poor Ground Conditions	IV Conventional Cut-and-Fill With Good Ground Conditions	V Back Stope Cut-and-Fill With Good Ground Conditions	VI Conventional Cut-and-Fill With Good Ground Conditions	VII Conventional Cut-and-Fill With Good Ground Conditions	Qualifications
Development (Figs. 1a and b and 2)									8-hr shift, 6.5 hr effective time at face
Case A, 7-ft vein, $2.34	2.34	2.34	—	—	—	2.22*	—	—	Labor rate $10.50 per hr, includes bonus and fringe.
Case B, 7-ft vein, $2.65	2.65	—	2.65	2.65	—	—	—	—	
Case C, 7-ft vein, $2.29	2.29	—	—	—	2.29	—	—	—	
Case D, 7-ft vein, $0.96	0.96	—	—	—	—	—	0.96	0.96	
Drill and Blast									Unit operation costs include labor.
Breasting 7-ft vein, 20.3 st† jackleg	2.67	2.67	—	2.67	2.67	—	—	—	
Backstope, 7-ft vein, 90.6 st stoper	1.69	—	1.69	—	—	—	—	—	Equipment costs include maintenance parts and labor, tires, and fuel. Equipment depreciation is not included.
Backstope, 7-ft vein, 129.4 st jumbo	1.81	—	—	—	—	1.81	—	—	
Breasting 18-ft vein, 93.2 st jackleg	1.38	—	—	—	—	—	1.38	1.38	
Mucking									
Single slush 25 hp†	1.10	1.10	1.10	—	—	—	1.10	—	
Double slush 25 hp	2.20	—	—	2.20	—	—	—	—	
Cavo 511, pneumatic	0.70	—	—	—	0.78	—	—	0.78	
LHD 1 cu yd diesel	0.92	—	—	—	—	0.92	—	—	
Ground Support									All materials are based on current 1976 prices.
Rockbolts, 7-ft vein	2.46	2.46	2.46	2.46	2.46	2.46	—	—	
Rockbolts, 18-ft vein	1.47	—	—	—	—	—	1.47	1.47	
Stulls, 7-ft vein	0.71	—	—	0.71	—	—	—	—	
Backfill									Hydraulic sand fill taken to be $1.25 per st ore extracted.
Gob	1.47	—	—	1.47	—	—	—	—	
Hydraulic sand	1.25	1.25	1.25	—	1.25	1.25	1.25	1.25	Machine ownership costs excluded.
Stope preparation									
Liner place, carried blind	1.65	—	—	—	1.65	—	—	1.65	
4-compartment timber, carried blind	1.72	1.72	—	—	—	1.72	—	—	
4-compartment timber, chutes carried blind	0.79	—	0.79	—	—	—	—	—	
Combined timbered	2.51	—	—	2.51	—	—	2.51	—	
Total cost per st		$11.54	$9.94	$14.67	$11.10	$10.38	$8.67	$7.49	
Stope crew size		2	2	2	2	2	2	2	
Productivity (st per mine shift)		12.7	21.2	7.0	15.6	30.4	13.8		

*One chute installation eliminated to allow a 91.4-m (300-ft) optimum LHD haul length.
†Metric equivalents: 1 ft × 0.3048 = m; 1 st × 0.907 184 7 = t; 1 hp × 745,699 9 = W.

FIG.10.29 (Pugh & Rasmussen, 1982)

Pugh and Rasmussen also studied the costs and productivities of the highly mechanized ramp-in-stope cut-and-fill system shown in Fig. 10.18. The costs are summarized in Fig. 10.30 (including the detail of the costs for each of the two blocks, or stages, into which the stope has been divided). The costs are in 1977 dollars as well. The productivity estimates are shown in Fig. 10.31. Note the lower costs ($5.72 per ton) and the high productivities.

Summary of Mining Costs

	Cost, $	Minable† Tonnage	Cost Per Ton,‡ $
First block	639,343	92,069	6.95
Subsequent block	464,186	86,262	5.38
Final block	308,375	68,501	4.50
Total	1,411,904	246,792	5.72

*Excluding sill and crown pillars to give a recovery of 89.4% of reserves in place.
†Operating costs only. Does not include amortization of capital, etc.
‡Metric equivalent: st × 0.907 184 7 = t.

FIG.10.30 (Pugh & Rasmussen, 1982)

Summary of Productivity Estimates for Highly Mechanized Cut-and-Fill Stoping

Calculations of the number of men required to produce 800 stpd*

Backstoping
200 st per shift per stope is achievable on a continuous basis
Stope crew = 1 jumbo operator
          1 miner
          1 LHD operator
          _____
          3 per shift

Assume 2 stopes each being worked on 2 shifts per day to give 800 stpd
          12 men per day + 4 utility crew (stope preparation, salvage, piping, etc.)
Total =   16 men per day
Backstoping productivity = $\frac{800 \text{ st}}{16 \text{ men}}$ = 50 st per miner shift

Breast Stoping
Assume 2 men per stope per shift will produce 100 st (2 rounds per shift) so that 4 stopes worked on 2 shifts per day basis produce 800 st.
Stope crew = 1 jumbo operator
          1 LHD operator
          = 16 men per day + 8 utility crew
          = 24 men per day
Breast stoping productivity = $\frac{800 \text{ st}}{24 \text{ men}}$ = 33.3 st per miner shift

*Metric equivalent: st × 0.907 184 7 = t.

FIG.10.31 (Pugh & Rasmussen,1982)

A big advantage of cut-and-fill over sublevel stoping, the most frequent alternative method, is its low dilution. This factor (the low dilution of cut-and-fill) becomes important when evaluating the economics of cut-and-fill against other mining methods, particularly for rich and valuable ores. At Boliden (World Mining, October 1982) such a comparison has been done. The conditions for the calculations were : mining cost for blasthole stoping of 75% that of cut-and-fill, and waste dilution of 20% for blasthole and 10% for cut-and-fill. In Fig. 10.32 it can be seen that if the ore value is greater than 375 Swedish Crowns (US$50) per undiluted ton of ore, the cut-and-fill method gives a better result than blasthole stoping (sublevel stoping). In summary, when evaluating cut-and-fill against other mining methods, it is not just the cost per ton that should be considered but the cost per ton of metal. (The previous relationship could be related to the old thumb rule advising "not to dilute what is already concentrated"). In some circumstances, when using the cost per ton of metal to compare two alternative methods, these figures may decisively favour the apparently costlier mining method.

Comparative profitability

FIG.10.32
(World Min.,
Oct.1982)
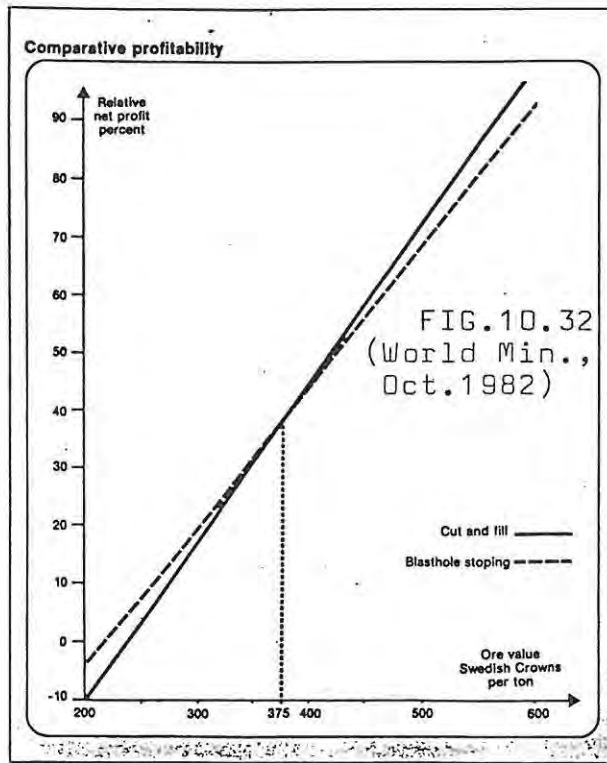
11. CONCLUSIONS

The first and most important conclusion is that the ultimate purpose of any mining venture is to maximize its profits. Other important constraints, frequently enforced by the local governments, include the desire of maximizing the ore recovery, maximizing the productivity (to afford the increasing costs of labour), and to provide safe working conditions. Any mine plan should try to attain these objectives. They control every aspect of the mining process: cutoff grade determination, delineation of the orebodies, selection of a mining method and mining schedules, choosing a mine-mill rate, level of expenditure in exploration and in other geological investigations and services, etc.

The increasing stripping ratio is the main limitation to the application of open pit mining. The stripping ratio is, in turn, controlled by the shape, size and orientation of the orebodies, by the slope stability, and by the depth of mining.

The application of caving methods is dependant upon the shape and sizes of the orebodies (horizontal sections large enough to achieve caving when undercut, and vertical extensions large enough to carry the costs of development are prerequisites), the homogeneous distribution of the values within the orebodies, and the cavability of the orebodies (block caving) or of the hangingwalls (sublevel caving). A good draw control is essential to cut dilution and to increase the recovery of the ore. A model of gravity flow of the broken ore in sublevel caving has been developed and tested, and relationships useful to layout planning and draw control have been derived from this model. In block caving, the "angle of retreat" is a useful concept in draw control. In both methods "draw charts" are widely used to implement the desired grade control policies.

The fairly regular nature of the orebody contacts, its steep dip to achieve gravity flow of the ore once it has been blasted, and the competency of the orebody and especially of the wall rock to cut dilution, are the main limitations to the application of sublevel stoping. The mining layout must be carefully planned in order to achieve selectivity

(which is not very high with this method),   maximizing the ore recovery and cutting down dilution; an understanding of the blasting techniques is necessary for an optimum layout planning and grade control.

The room-and-pillar mining method, used in flat dipping tabular orebodies, is seriously affected by the rock mechanics of the hangingwall. A good understanding of this factor, and in particular of the process of arching taking place above the stope, will permit the optimization of the pillar support system, thus maximizing the ore recovery.

In shrinkage a good downwards flow of the broken ore is an essential condition, and the sidewalls must be fairly competent to avoid excessive dilution during the final draw.

The possible applications of cut-and-fill mining cover a very wide range indeed.   The related geological work include a moderately detailed delineation of the orebody to produce a general mine plan (particularly of the accesses) and stoping schedule, and constant supervision of the stoping (including "secondary" sampling during the stoping) to ensure a maximum ore recovery with minimum dilution.

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