DESIGN OF SUB LEVEL CAVING METHOD
BY MEANS OF MINE MODEL TESTS

By

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We accept this thesis as conforming to the
required standard.

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ABSTRACT

Design criteria of the Longitudinal Sub-Level Caving method of mining, as it would particularly apply to Granduc Mines, has been studied by means of geometrically scaled mine models.

Principles of similarity and their use in the model test work with due consideration to the simplifying assumptions have been analysed. Gravity flow theories of granular material as applicable to the bin design work have been used where possible in the stope design.

A total of thirty-seven tests on a 1:30 scale model were conducted involving various orebody configurations. Qualitative observations are reported and the effects that will have to be dealt with by theoretical treatment are described. Within reasonable experimental accuracy, the draw figures of the flow of broken ore material in the model are determined. Based on these figures, mine layout patterns which would ensure maximum ore recovery with minimum waste dilution from the stopes have been presented for the mine development work.

Quantitative design of 'blast retreat distance' depending on change in natural conditions of the stope, such as moisture content and confining pressures, etc. have been determined by measuring the flow properties of the ore material with Triaxial compression testing equipment. A remarkable change in the flow properties and hence the draw configuration is predicted.

Recommendations on further and advanced work are included on the quantitative design of stoping layouts for the modern sub-level caving methods.
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April, 1970

STEWART, B. C.
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NOMENCLATURE

\( \alpha \) - Ring gradient

\( a_N \) - Semi-major axis of the ellipsoid of motion

\( A \) - Area of the opening

\( A_w \) - Width of slice

\( b_N \) - Semi-minor axis of the ellipsoid of motion

b.r.d. - Blast retreat distance

\( B \) - Width of the extraction drift

\( C \) - Size of the extraction area

\( d \) - Average size of particles

\( d_d \) - Digging depth of the scoop

\( ds \) - Spherical diameter of particles

\( D \) - Diameter of largest lumps in blasted ore

\( D_h \) - Hydraulic or perimetral diameter =

\[
4 \times \frac{A}{\text{perimeter of orifice}}
\]

\( e_N \) - Eccentricity of the ellipsoid of motion

\( E_N \) - Volume of the ellipsoid of motion

F.W. - Foot wall

\( \gamma \) - Unit weight of granular material

\( g \) - Acceleration of gravity

\( h \) - Height of extraction drift

\( h' \) - Height of the gravity flow

\( h_N \) - Height of the ellipsoid of motion

H - Head of packing above opening

H.W. - Hangingwall

K - Properties of the lumpty material
$K_N$ - Volume of discharge

$l$ - A significant distance

$\lambda_i$ - Any pertinent distance

$n$ - Length scale

$\theta_r$ - Ore recovery = % ore recovered to ore blasted

$P$ - Width of pillar between drifts

$\varphi$ - Angle of internal friction

$R_d$ - Blast retreat distance (single or multiple ring burden)

$\rho_s$ - True density of solids

$\rho_b$ - Bulk density of packing

S.L.I. - Sub level interval (or 'S')

S.F. - Swell factor

\[
\text{Swell factor} = \frac{\text{Volume of solids} + \text{volume of voids}}{\text{Volume of solids}}
\]

$\psi$ - Natural angle of repose

$\sigma$ - Normal stress on the failure plane

$\sigma_{hv}$ - Horizontal stress on the vertical surface of sliding

$\sigma_v$ - Vertical stress

$\theta$ - Angle of side slopes

(Angle of inclination of the hopper bottom)

$\tau$ - Shear stress on failure plane

$T_e$ - Total extraction =

% of rock loaded to ore blasted (100%)

$v$ - Velocity of discharge from opening

$V$ - Draw volume

$V_C$ - Volume of container
$V_N$ - Volume of discharged material

$\overline{V}$ - Average vertical pressure

$w$ - Specific weight of the material or dead weight (Dimensionally, FL$^{-3}$)

$W_d$ - Waste dilution =

% of waste loaded to total rock loaded (100%)

$Z$ - Depth below the surface
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1.1 Location and Geology:

The Granduc Mine is near the British Columbia - Alaska boundary in rugged mountainous country, about 36 miles Northwest of Stewart, B.C.

Access to the mine from Stewart is made by means of a 32 mile road to Tide Lake Camp, from where an 11 1/4 mile long tunnel connects with the mine. The Concentrator is located at Tide Lake Camp and the townsite is situated at Stewart.

Forty-three million tons of ore, averaging 1.73% copper before dilution, are reported to have been indicated by diamond drilling (1966). The deposit is classified as Mesothermal Replacement. The orebodies occur in folded and faulted siliceous metasediments cut by feldspar porphyry dikes. Mineralization consists essentially of pyrrhotite, chalcopyrite, pyrite and sphalerite.*

1.2 Purpose:

This thesis is concerned with the design of the Longitudinal Sub-Level caving method, particularly with respect to the Granduc Mines. Sub-level caving, with trackless equipment, has been chosen as the mining method. The principles for determining the best method for mining the Granduc orebodies have been based on high productivity (7,500 tons/day) and low cost, together with the acceptable premise of good mining practices, which include the desirability of maximum recovery and minimum dilution.

In the transverse as well as in the longitudinal sub-level caving, the ore is longhole blasted, whereas the waste is caved. Test drifts driven

* See Appendix VI for detailed geology.
into the hanging wall, which is mainly in the sediments, shows a marked degree of cross-fracturing which renders it easily fragmentable and liable to cave. Furthermore, there occurs a major fault within ten to twenty feet away from the hanging wall contact. In the worst case, induced caving may be required.

The wider portions of the orebody, which are defined here as anywhere between fifty and one hundred and twenty feet, are planned to be mined by the transverse sub-level caving method (Footwall to Hanging wall retreat). This has been primarily on the recommendations of the consultants as well as on some field experience obtained elsewhere in Canada and abroad. The wings of the orebodies are narrow and vary in widths from fifteen to fifty feet on the whole; approximately more than 50% of the total mineable tonnages are tied up in these narrow widths, which call for mining methods other than the transverse sub-level caving. Therefore, it became imperative for Granduc Mines to conduct a study program to determine the best method of mining of these areas. Longitudinal sub-level caving method (retreat along the strike) is considered most suitable in this case. The principal reason being that the same equipment and the basic development work as is needed for the mining of adjacent transverse sub-level stopes could be used. Preliminary planning of these areas has been done but not with much exactitude, since this field is relatively unexplored. Further research work was needed to establish the optimum layouts, hence this study was taken up. The importance of these tests, locally performed, is further emphasized by the fact they serve as a constant source of reference for mine planning and layout work. Also the presence at the mine of permanent records of pictures, slides and other visual aids developed during the course of testing, are useful for the training of the mine personnel from time to time.
1.3 Scope:

A 1:30 scale mine model was constructed at the property and tests were conducted on a carefully drawn up test program. Optimum layout of the production drifts/draw points on proper sub-level interval depends on the study of the "cave figures" or the "ellipsoid of motion" of the broken rock in the stopes. Therefore, actual ore from the mine crushed to size for the model work was used to determine these figures. The testing was divided into two areas of study:

(a.) The first series of tests were carried out to check the figures which have already been used for the planning work of the transverse sub-level caving methods for the wider orebodies. This has a special importance, for a comparison of the model work could be made with the actual performances as some operational experience is obtained at a later date.

(b.) The second series of tests were performed on the longitudinal sub-level caving method. With the help of the Geological Department, a table was prepared which showed the widths of the orebodies, with their associated footwall angles and tonnages. This helped in programing the test work in order of importance of these areas. Tests representing orebody widths of 20', 30', 40', and 50' at various footwall angles and with change in parameters such as sub-level interval, location of extraction drifts and footwall slash, etc., were carried out in detail.
Many preliminary tests were needed before a satisfactory testing procedure could be developed. A discussion on some of the problems encountered has been included in Chapters 3 and 5.

Additional testing was done to determine the change in flow properties of the broken ore over a range of moisture contents by using a Triaxial Compression testing equipment. The results are included in Chapter 5.

Proposed mine layouts based on the test work are presented in Appendix IV, and suggestions on further and advanced work are included in Chapter 7.
2.1 **General:**

Sub-level caving in various patterns of blasting and extracting was the earliest caving method used in mines with weak and incompetent ground conditions; in spite of a high dilution and relatively low recovery, it was regarded as the only practical method available. With mechanized development and long-hole drilling equipment, improved blasting techniques and the advent of trackless extraction units, this method is being increasingly considered and applied in base metal and ferrous mines with large, thick and competent orebodies. This is because it can be readily mechanized, it is safe, standardization of equipment presents fewer problems, selectivity in variable-grade orebodies is possible and a large proportion of the development is in ore. These factors all contribute to keeping costs down and the method is now often preferred to block caving or large sub-level stoping.

Due to the inherent characteristic of sub-level caving, in that the thin column of ore drawn with each fan blast is bounded by the solid wall of the next fan on one side and the caved waste on all the other sides, the problems of recovery and dilution are as great as in any other method. During the last ten years or so a considerable amount of research has been carried out in many countries, both in the form of scale-model tests and of studies conducted under natural conditions. This research has shown that by introducing improved layouts based on the draw characteristics and employing modern mining techniques, a good recovery in conjunction with a dilution comparable to that obtained with a carefully controlled block cave method is possible. It has been indicated, however, especially in base metal ores, that a strict positive draw control is essential to the success of the
The term 'positive' in this context means obtaining a comprehensive draw grade and extraction tons record to stop the draw at the desired point and also to gain reliable recovery and dilution figures in each drawpoint.

2.11 Theory of Sub-Level Cave Draw:

So far as is known, the bulk of research into the sub-level caving mining method has been carried out in Sweden and, in particular, by the Royal Institute of Technology, Stockholm. Janelid and Kvapil's (47) paper on the principles and theory of sub-level caving is probably the most authoritative to date.

Model experiments and underground studies are known to have been carried out in Australia at Mount Isa Mines, in Zambia at Mufulira Copper Mines, in Sweden at Kiruna and Malmberget Mines, and in Canada at Craigmont Mines and Frood Stobie Mine.

It is not possible here to expound all the characteristics involved in sub-level caving draw, but those that have a major influence on the method of draw control, and therefore, on the understanding of it, are described.

2.12 Shape of Draw:

Basically, it has been found that the gravity flow of blasted or caved rock, under pressure, obeys the same laws and is approximately similar to the flow of granular material in a bunker. Broken rock entering a drawpoint flows from a zone in the shape of an ellipsoid. The actual ground drawn flows from what Janelid calls the 'ellipsoid of motion'. This, in turn, disturbs a larger zone, called the 'limit ellipsoid', which is approximately fifteen times the size of the ellipsoid of motion (see Fig. 1A). If the
material being drawn consisted of laminations of different colours, and could be viewed through a glass plate, the draw would appear to be in the form of a cone with the apex at the drawpoint and the sides slightly concave (see Fig. 1B).

In Fig. 1E. this ellipsoid of motion is marked $E_N$. Distinction can be made between its semi-major axis $a_N$ and its semi-minor axis $b_N$.

The volume of the ellipsoid of motion $E_N$ approximately corresponds to that of the discharged material $V_N$ (Fig. 1D) and to the volume of the discharge cone $K_N$ (Fig. 1C). Therefore, the relationship between these can thus be described by:

$$E_N \sim K_N \approx V_N$$  \hspace{1cm} 2 (1)

If the volume $V_N$ and the height of the ellipsoid of motion are known, the semi-minor axis $b_N$ of Fig. 1E can be calculated from the formula:

$$b_N = \sqrt{\frac{V_N}{2.094 h_N}}$$  \hspace{1cm} 2 (2)

The characteristics of the shape of the ellipsoid of motion are determined by its eccentricity $e_N$

where

$$e_N = \frac{1}{a_N} \left( a_N^2 - b_N^2 \right)$$  \hspace{1cm} 2 (3)

The terms $a_N$ and $b_N$ in equation 2 (3) correspond to those of Fig. 1E.
FIG. IA GROUND UNDER PRESSURE IS DRAWN FROM A ZONE IN THE FORM OF AN ELONGATED ELLIPSOID, TERMED THE ELLIPSOID OF MOTION. THIS DISTURBS THE GROUND IN A LARGER ZONE, TERMED THE LIMIT ELLIPSOID, FROM JANELID KVAPIL.

FIG. IB RELATIONSHIP BETWEEN VISIBLE DRAW CONE AND ELLIPSOID OF MOTION AND LIMIT ELLIPSOID. MATERIAL DRAWN FROM B IS REPLACED BY MATERIAL GRAVITATING FROM AREAS A AND C. FROM JANELID AND KVAPIL.

FIG. IC, ID, IE

ZONES OF MOTION DURING RUN-OFF OF GRANULAR MATERIAL

FIG. IF, IG, I H, I I

FORM OF ELLIPSOID OF MOTION AS A FUNCTION OF PARTICLE SIZE.

FIG. I GRAVITY FLOW OF GRANULAR MATERIAL
The characteristics of the shape of the ellipsoid of motion, i.e., its eccentricity, are not constant, but depend very much on the particle size of the material. Smaller particles of material correspond to a slimmer ellipsoid of motion and to a greater eccentricity. Larger particles extend the ellipsoid in width and its eccentricity becomes less. This is shown schematically in Figs. 1 (F, G, H).

For the same material, the eccentricity depends on a number of factors such as the size of the discharge opening (enlargement of the discharge opening increases the eccentricity), the height of the ellipsoid of motion (a greater height increases the eccentricity), the velocity of discharge (a faster rate of discharge increases the eccentricity), etc.

The laws in connection with gravity flow of granular material do not undergo any basic changes, even when the gravity flow is prevented for various reasons from developing fully and symmetrically to the vertical axis. Such cases arise, for example, if the discharge opening lies not in the centre of the bunker bottom, but in the side wall (see Fig. 11).

The foregoing principles have been used by Janelid and Kvapil and others in determining the parameters of particularly the Transverse Sub-Level Caving method.

2.2 **Principles of Gravity Flow and Design of Mass-Flow Bins:**

Principles of gravity flow have been applied in the quantitative design of the bins for some time now. Many references on the subject were reviewed with the intent of studying applications in stope design. Some of these are mentioned here.

Since the classical work of Janssen (44), Ketchum (43) and Jamieson on the gravity flow of grains in elevators and bulk solids in bins,
several writers have discussed this problem. For example - Kvapil (13, 14), Bernache (15), Mroz and Drescher (16), Roberts (17), Peschl (19), Aytaman (22, 23, 24), Handley and Perry (31), Jenike 1, 3, 4, 5, 6, 36, 37, 38), Johnson (7, 8, 9), Walker (10, 11) and others. At the present time, there are on record, more than 200 selected references on the subject of Flow of Granular Materials.

However, by far the most important contributions to the study of the application of the principles of gravity flow in the design of bins, has been made by Jenike, Johnson and Walker. Therefore, without going into any detail of the works of all the writers at this stage, works of only the last three authors mentioned above have been discussed here.

The development of a mathematical theory of gravity flow of such solids as Ores, Concentrates, Coal and Chemicals, etc., by Jenike (37), is based on the concepts of Soil Mechanics and Plasticity. While the concepts of this theory were formulated by him in 1953, most of the work was carried out at the bulk solids Flow Laboratory (1957 - 1962), of the Utah Engineering Experiment Station, University of Utah.

Jenike (3), has stated that once flow properties of solids have been measured and are known, then the critical dimensions of the hopper walls can be determined. It is well known now, from the works of various authors, that flow properties of solids vary, for instance, due to changes in moisture content, size distribution, shape, bulk density, angle of internal friction and temperature, etc., The conditions of lowest flowability have to be used in the calculation of critical dimensions.

A direct-shear, constant rate of strain machine is employed as a measuring unit. Specimens of the tested solid are first consolidated within
the shear cell and then sheared to obtain a relation between the consolidating pressure and the resultant strength of the solid. This gives the principle flow properties of the bulk solids, namely, the flow-function. The tester also measures the angle of friction between a solid and samples of wall materials as well as other properties used in the design of flow. Flowability tests can be carried out for a range of moisture content, to determine the maximum moisture for which a solid can still be handled by gravity.

Walker's (10, 11) work was stimulated by the difficulties experienced by Central Electricity Generating Board, Bristol, England, in getting wet fine coal to flow through the bunkers, hoppers and chutes, etc. Most of his work was carried out between the period 1961 through 1967. The approach adopted to obtain the design data for bunker flow is basically similar to that adopted by Jenike. However, Walker (11), has simplified the calculation to derive the approximate stresses that would occur within a granular material flowing in a bunker and has claimed that his use of the ring shear tester (12), compared to the direct shear tester of Jenike, has proved more agreement between the theory and the practical design.

In either case, model tests have been carried out on hoppers designed by Jenike's and Walker's method. Material is tested and a container is built that ensures the mass-flow.

It would be useful to define here, two types of flow patterns; mass-flow and funnel flow or plug flow. In mass-flow bins, all the solid is in motion whenever any of it is drawn out. In a funnel-flow bin, flow occurs within some channel surrounded by non-flowing (dead) solid. In a mass-flow bin, in which the flow channel coincides with the bin itself and, hence, is defined and constant, bin loadings are also well defined and reproducible.
In a funnel-flow, it forms within the solid itself. The channel may expand or contract as the flow properties of solids change with varying moisture content and fragmentation, etc. As a result, loading in funnel-flow bins is more erratic and less reproducible than in mass-flow bins. In addition to the mass-flow bins and funnel-flow bins, Jenike (3) has described another type, i.e., expanded-flow bins, where use is made of a short mass-flow hopper unit under a funnel-flow bin. The mass-flow hopper serves to expand the size of the flow-channel to a dimension which eliminates the possibility of rat holing. It can be added here that the design of Transverse Sub Level Caving or Longitudinal Sub Level Caving for wide ore-bodies, can make a special case of a series of mass-flow hoppers under a funnel-flow bin.

Handley's (31), experimental results for sand in the converging sections of 65° and 70° hoppers, has shown similarity to the results predicted by the more rigorous stress solution of Jenike, Johnson and Walker. Calculated pressures of the same order of magnitude, were obtained as actually measured. The close similarity only applies very near the hopper outlet and the importance of pressures in this region is emphasized in order to find whether mass-flow will be fully developed. A small pressure sensitive radio pill was used for the measurement of internal stresses during flow.

It is clear that the conditions in a sub-level caving stope, are different from that of a bin and, therefore, the application of the principles of quantitative design of bins to be used for stope design have only a limited application.

2.3 Principles of Similitude in the Flow of Granular Materials:

Another topic of particular interest to this project, is the principles of similitude so that the behaviour of a prototype may be correctly
predicted from the observations on a model. For this to be achieved, it is necessary to assess not only the physical quantities that are relevant to the problem, but also to use judgement to reduce them to a working minimum by selecting the most significant parameters.

Applications of dimensional analysis to model testing, have been described by various authors, but only the important ones are mentioned here.

Roscoe (25), has shown the use of dimensional analysis for model testing in soil mechanics to a certain degree. Fowler (28), in his paper, "The Flow of Granular Solids Through Orifices", has derived the prediction equations of the weight discharged per unit time, for granular materials through orifices. He has emphasized the importance of the use of bulk density over the true density of materials and, also the shape factor of the particles considered in his analysis.

Matthee (26), studied the segregation phenomenon relating to bunkering of bulk materials with the help of geometrically reduced model bins and has discussed the scale up of these results. To reduce the number of experiments, he introduced dimensionless groups and included variables like grain size, grain size distribution, particle shape, particle density, flowability, angle of repose, resistance to agglomeration, surface characteristics, the shape and diameter of bunker and its orifice, the height of material in the bin, wall friction, the method of feeding and discharging and others. He reports that, for a solution of the problem of segregation, all the variables that influence segregation and the equations of motion of particles, need to be known. Such a solution is not yet possible.

In "The Behaviour of Granular Materials in Flow-out of Hoppers" by Reisner (27), are listed the factors influencing the efflux as used in the
formulae of various authors. They are, for instance, diameter of the bin, diameter of the outlet, grain size, reduced area of efflux, height of the surface of the stored material in the bin, area of efflux, cohesive forces, hydraulic radius, acceleration of gravity, bulk density of the stored material, internal friction, angle of inclination of the hopper bottom, specific weight, shape factor, true flow factor and specific pressure on the bottom of the bin. It is noted that no single author has considered more than a handful of variables and a comprehensive approach is lacking in the literatures, therefore, it can be concluded that there is no formula so far, which can be used without any restriction for all types of bins and all kinds of bulk materials.
CHAPTER 3
THEORETICAL CONSIDERATIONS

3.1 General:
A discussion on the design of sub-level caving stopes, from the model tests, must inevitably include the aspects of the theory of model testing, the applications of the gravity flow principles to the granular materials and their arching phenomenon, etc.

3.2 Theory of Models:
An attempt has been made to apply the theory of models, to get an indication of the relationship of the variables taking part in the phenomenon so that the observations made on the mine model may be used to predict the performance of the prototype, i.e., the stope. The pertinent variables considered in this analysis, along with the simplifying assumptions, are listed in Appendix 1.

Equation (5) of Appendix 1 is rewritten in the dimensional form as:

\[
\frac{V}{13} = F \left( \frac{A_i}{1}, \frac{A}{12}, \frac{Dh}{1}, \frac{w.1}{\rho_s v^2}, \frac{d}{1}, \frac{H}{1}, \frac{P^b}{\rho_s}, \frac{Vc}{1}, \frac{dd}{1}, \frac{\theta}{1} \right)
\]

A similar equation may be written for the model:

\[
\frac{V_m}{13_m} = F \left( \frac{A_{im}}{1m}, \frac{A_m}{12_m}, \frac{Dhm}{1m}, \frac{Wm.1m}{\rho_{sm} v^2_m}, \frac{dm}{1m}, \frac{Hm}{1m}, \frac{P_{bm}}{\rho_{sm}}, \frac{Vcm}{13_m}, \frac{ddm}{1m}, \frac{\theta}{1m} \right)
\]

Since each equation refers to the same type of system, the functions are identical in form.
Now the "Length Scale" is defined as the ratio of some pertinent distance or length of the prototype of the corresponding distance in the model and is designated as \( n \), or

\[
1 = n l_m
\]

in the case of the model used in this analysis, \( n = 30 \). From the above, the design conditions for the model may be determined as follows:

\[
\frac{V_m}{1^3_m} = \frac{V}{1^3} \quad \text{or} \quad V_m = \frac{V}{n^3}
\]

In other words, volume in the bucket drawn from the model is \( \frac{1}{(30)^3} \) times volume of the actual bucket used in the mine.

Similarly, the other design conditions are derived:

\[
\frac{\lambda_{im}}{\lambda_{im}} = \frac{\lambda_{i}}{1} \quad \text{or} \quad \lambda_{im} = \frac{\lambda_{i}}{n}
\]

\[
\frac{A_m}{A_{m}} = \frac{A}{1^2} \quad \text{or} \quad A_m = \frac{A}{n^2}
\]

\[
\frac{D_{hm}}{D_{h}} = \frac{D_{h}}{1} \quad \text{or} \quad D_{hm} = \frac{D_{h}}{n}
\]

\[
\frac{w_{m}}{w_{m}} = \frac{w_{m}}{1} \quad \text{or} \quad w_m = \frac{n.w. \rho_{s_m}}{\rho_{s}.v^2_m}
\]

\[
\frac{d_{m}}{d_{m}} = \frac{d}{1} \quad \text{or} \quad d_m = \frac{d}{n}
\]

\[
\frac{H_{m}}{H_{m}} = \frac{H}{1} \quad \text{or} \quad H_m = \frac{H}{n}
\]

\[
\frac{\rho_{bm}}{\rho_{s_m}} = \frac{\rho_{b}}{\rho_{s}} \quad \text{or} \quad \rho_{bm} = \frac{\rho_{b}.\rho_{s_m}}{\rho_{s}}
\]

\[
\frac{V_{cm}}{V_{c}} = \frac{V_c}{1^3} \quad \text{or} \quad V_{cm} = \frac{V_c}{n^3}
\]
Design Equation 3 (1b) - Since $\lambda$ refers to every dimension, this equation imposes the condition of geometrical similarity in all respects between model and prototype. Both as to form of the model, cross sections at all points, and loading pattern. Hence, the model was designed according to this.

Design Equations 3 (1c), 3 (1d), 3 (1f), 3 (1g), 3 (1i), and 3 (1j) indicate that the design conditions regarding the area of opening, hydraulic diameter of the opening, size of the particle, head of packing above opening, volume of container and the digging depth of the scoop give complete freedom in the selection of the length scale between the model and the prototype.

Design Equation 3 (11) - is satisfied because the same slope angles near the opening have been used as that in the mine.

Design Equation 3 (1k) - is satisfied because actual ore from the mine is used in the model. The crushed rock for the model (sizes down to $1/4"$) generally resembled in shape, the broken rock underground. No effect of moisture has been considered in the analysis.

Design Equation 3 (le) - Here $P_{sm} = P_s$

Therefore, $w_m = n.w. \left( \frac{v_m}{v} \right)^2$ - 3 (2)

From Fowler (28), approximate velocity of discharge $v$ from the orifice is given by:

$$v = 0.236 \times \sqrt[4]{2g D_h} \times \left( \frac{D_h}{d_s} \right)^{0.185} \text{ cms/sec}$$ - 3 (3)
Therefore, from equations 3 (2) and 3 (3):

\[ w_m = n \cdot w \cdot \left( \frac{0.236 \times \sqrt{2 \ g \ \text{m} \ Dh_m}}{0.236 \times \sqrt{2 \ g \ \text{h} \ Dh}} \times \left( \frac{Dh_m}{ds_m} \right) \right)^2 \]

Since the value of 'g' is the same for the model and the prototype - there, fore, by solving Equation 3 (4), we obtain:

\[ w_m = n \cdot w \cdot n^{-1} \]

or:

\[ w_m = w. \]

Therefore, Equation 3 (1e) is satisfied within practical limits since Equation 3 (3) allows the prediction of flow rates through orifices with an overall accuracy of + 10 percent.

Design Equation 3 (1h) - indicates that corrections are needed to be applied to the draw volumes - \( \rho_s^m = \rho_s \), but the exact nature of \( \rho_b \) is not yet known. To apply corrections or to determine the extent of distortion in the results of ore recovery and total extraction obtained from the model tests, it is imperative to find the swell factor, thereby the bulk density \( \rho_b \), of the broken column of ore in the stope. These measurements are not possible at this stage until a working stope is available for such tests.

In conclusion it can be said that, due to difficulties in simulating consolidation effect of blasting in the stope and the density difference between the mine ore and the model ore due to negligible expansion of the
blasted ore in the mine, right conditions are not represented in the model.
It is clear, however, that tests with crushed ore give qualitative results
which are not necessarily higher or better than corresponding results from
the mine. So it is justified to carry out this type of testing.

3.3 Gravity Flow and Determination of the Parameters in Sub-Level Caving:

The gravity flow of material in sub-level caving corresponds in prin­
ciple to the case of Figure 1 (1) because the gravity flow in the sub-level
caving is, figuratively speaking, cut-off by the wall of the slice.

Typical sections in the Case of Transverse and Longitudinal Sub-Level
Caving method are shown in Figure 2 and Figure 3, respectively.

The sub-level caving method is characterized by a gravity flow of
lumpy material because both the blasted ore and the waste may contain large
lumps of over 16 inches or so.

To optimise the production of clean ore and minimize contamination
by waste as well as ore losses, it is necessary to determine the optimum
parameters of the slice. This is mainly a matter of determining the sub­
level interval $S$, the blast retreat distance $R_d$, the width of slice $A_w$, the
ring gradient $a$, the width $B$ and height $h$, of the extraction drifts and the
width $P$ of the pillars between drifts.

The precise optimum values of the parameters of sub-level caving,
including all detailed effective factors, can only be found on the basis of
the tests under natural conditions. However, Janelid's basic formulae
(Equation 2 (2) and Equation 2 (3) give approximate figures for the para­
parameters of sub-level caving which can be applied in planning the mining system

The above are directly applicable in the design of Transverse sub­
level caving where the draw figures are symmetrical solid geometrical shapes
FIG. 2 A  DIAGRAMMATIC VERTICAL SECTION THROUGH LONGITUDINAL AXIS OF EXTRACTION DRIFT

FIG. 2 B  DIAGRAMMATIC VIEW OF THE SLICE WALL IN THE PLANE OF THE SECTION 1-1 OF FIG. 2A

FIG. 2  TRANSVERSE SUB LEVEL CAVING
FIG. 3A Diagrammatic vertical section through longitudinal axis of extraction drift.

FIG. 3B Diagrammatic view of the slice wall in the plane of the section J-J of FIG. 3A

FIG. 3 Longitudinal sub level caving
but in the case of Longitudinal Sub-Level Caving draw figures are not symmetrical because of the almost parallel, inclined footwall and hangingwall of the stope. Therefore, draw figures are to be determined for each different configuration.

Considerations given to the design of parameters such as fragmentation, drift width, drift height, flow throat, the digging depth of the scoop and the ring gradient, etc. can be common to Transverse as well as Longitudinal Sub-Level Caving method, but the design criteria for the sub-level interval, pillar width, blast retreat distance, loading pattern and location of drift, etc. are different for each method.

Generally, Model testing was guided from the following considerations:

**3.31 Sub-Level Interval:**

The sub-level interval can be expressed by a symbolic function of the effective factors such as:

\[ S = f (K, \ h', \ c, \ v, \ R_d, \ x, \ m, \ B, \ P, \ z) \]  

Where \( K \) = Properties of the lumpy material which can be expressed in simplified form as the particle size

\( h' = \) Height of the gravity flow.

\( c = \) size of the extraction area.

\( v = \) velocity of discharge from the opening.

The other legends are the same as in Figure 2 and Figure 3.

The approximate sub-level interval in the case of Transverse sub-level caving is calculated from the formula:

\[ S \approx \frac{R_d}{\sqrt{(1-e_N^2)}} \]
Where \( R_d \) is the blast retreat distance (which can be a ring burden or a multiple of ring burdens).

The restraint on the sub-level interval of either 30' or 60' for the model test work was imposed, in order for it to be compatible with 30' sub-level intervals already accepted for the transverse sub-level caving of the 'C' orebody between the elevation of 3690' to about 2900', so that access to all the orebodies can be made with the same developmental work from the ramp system. There can be a possibility that with no such restraints as above, the optimum sub-level interval might exist between the figures of 30' and 60'. Sub-level interval of 60' and higher, however, have to meet other additional technical and economical requirements.

3.32 Digging depth of the Scooptram:

Janelid (47) has applied Rankine's theory on the distribution of the trajectories of the maximum principal stresses and calculated the optimum penetration of the scooptram into the slope. The trajectories of the principal stresses in the slope (Figure 3a) are inclined against vertical by 
\[
\frac{90° - \psi}{2}, \quad \text{where } \psi \text{ is the natural angle of repose.}
\]
The theoretical best depth \( x \) is given by the points 1 and 2 in the above figure.

In conformity with legends of Figure 2, theoretical best depth is calculated from the formula:
\[
x \approx h \cdot \cot \psi - h \cdot \tan \left( \frac{90° - \psi}{2} \right)
\]
taking \( \psi = 50° \) for Granduc ore. \( x = 5.16 \) ft.

The digging depth applied in practice should reach the theoretical figure. In the model tests, a digging depth of approx. 2 in. (2 in. x 30 = 5 ft.) was used.
Drift Width:

While planning the minimum width of the drift, fragmentation of the rock has been considered and used in the following formula from Janelid (47).

\[ B \geq \sqrt{\frac{5}{5D^2}} k \]  

Where \( D \) is the diameter of the largest lumps of the blasted ore and \( k \) is the factor of composition of the fragmented rock (used Nomograph from Janelid (47) P. 144.)

For the calculation of \( k \), used \( D = 16'' \) and assumed that fragmented rock constituted of lumps up to \( 16'' \approx 40\% \), medium size up to \( 20\% \), small size up to \( 40\% \). If percentage of fines and damp constituents such as powder, etc. is almost nil, then \( B \) calculates out to be equal to 12.0 ft. and if fines are considered up to \( 5\% \), then \( B \geq 15.5 \) ft.

16 ft. wide extraction drifts have been used for the planning of Transverse sub-level caving stopes, for equal loading from the sides with an 8 ft. wide scooptram. The minimum size of the drift used in the narrow and steep longitudinal stopes tests is 12 ft. x 12 ft. This conforms to the requirement of proper gravity flow and also permits the use of production drill jumbos within this dimension.

Location of Drift:

Theoretically, ideal layout for the transverse sub-level stopes is when width of drift = width of pillar.

This ensures nearly parallel gravity flow. However, due to practical difficulties, the best arrangement is to provide for side slopes.
In the case of longitudinal stopes, although it has been indicated before elsewhere that generally the ore losses along the F.W. can be reduced by locating the drift in the F.W., but it is clear that in order to locate the drifts most favourably, a study must be made for this mine, considering all factors fully such as the performance of the production drilling equipment, marginal values of the ore content in the F.W. at a particular location, and the rock stability problems. This last point is particularly important when extra wide drifts are needed, especially in the case of very flat dipping ore bodies to suit ring drilling.

3.35 Loading Pattern or Intensity of Loading:

The extraction width is given by the operating reach of the scooptram and by the loading system. The gravity flow approaches more closely a parallel form (and this is ideal in the case of Tr. Sub-level caving) if the operating reach of the loader is wider and the loading system covers the width of the extraction drift more fully - undesirable arching of the lumpy material is easily avoided this way.

When mining through longitudinal drifts in narrow deposits it is appropriate to draw more broken ore from the footwall side than from the hanging wall side. Loading at the footwall side should be increased, the smaller is the inclination of the deposit.

3.36 Ring Gradient:

Research in Sweden has indicated that the optimum ring gradient or the fan angle, within the angle limits of 60° - 120°, is mainly dependent on the ratio of average ore sizes to waste sizes. Janelid and Kvapil (47) produced a simple table which is reproduced, in a slightly modified form, below:
### Ratio of Rock Sizes:

<table>
<thead>
<tr>
<th>$K_O/K_W$</th>
<th>Fan Angle, $a$</th>
</tr>
</thead>
<tbody>
<tr>
<td>$&gt; 1$</td>
<td>Positive, $a &lt; 90^\circ$</td>
</tr>
<tr>
<td>$= 1$</td>
<td>Vertical, $a = 90^\circ$</td>
</tr>
<tr>
<td>$&lt; 1$</td>
<td>Negative, $a &gt; 90^\circ$</td>
</tr>
</tbody>
</table>

Where $K_O$ is the average particle size of the ore, $K_W$ is the average particle size of the waste and $'\alpha'$ is the angle between the horizontal and the fan, away from the direction of retreat (see Fig. 2 & 3).

The theory is that the ring gradient $'\alpha'$ is to have the effect of preventing as much as possible, the intermixing of waste or, inversely, the intermixing of ore into waste.

A material of fine particle size can, as a result of the gravitational force, fill the lower lying cavities and gaps in the material of coarse particle size, i.e., fine ore lying over lumpy waste or vice versa. The ring gradient is so chosen such that the optimum conditions are obtained in this respect.

The ring gradient is, however, also dependent on mining considerations -- such as maintaining a good brow and a good drilled footage efficiency. Probably because of these considerations, very few, if any, mines have adopted a backward sloping ring gradient or the fan angle and most have adopted angles varying from $70^\circ$ to $90^\circ$, depending on these conditions.

For the purposes of model test work, fragmentation of ore and waste were assumed to be equal and hence only vertical fans have been tested. This would need modifications, however, if in actual practice it is found that $\frac{K_O}{K_W} \neq 1$. 

3.37 **Blast Retreat Distance:**

From a theoretical point of view, the blast retreat distance $R_d$, will be governed by many factors. In principle, it can be expressed by a symbolic function of the following factors, where some of these may be interdependent.

$$R_d = f (K, h', c, v, S, x, m, B, \alpha )$$  \(3\) (9)

Where the first four factors are the same as used in Equation 3 (5) and the remaining factors as shown in Figures 2 and 3.

The proper blast retreat distance should, in the optimum case, correspond to the gravity flow of the blasted ore in the way shown in Figures 2a and 3a. The optimum blast retreat distance (See Fig. 2 & 3) for a vertical slice in a transverse sub-level stope can be calculated from the approximate formula.

$$R_d \geq S \sqrt{(1 - e^2)}$$  \(3\) (10)

Substitute value of $e = 0.975$, for a sub-level interval 'S' of 30 ft. from Janelid (47, Fig. 29, PP 142) in Equation 3 (10).

Therefore: $R_d \geq 30 \sqrt{1 - (.975)^2} \geq 6.60$ ft.

Blast retreat distances of 5, 6 and 7 ft. are tested in the model for the transverse sub-level caving method.

No formula is available for the calculation of blast retreat distance in the case of longitudinal sub-level caving method because of unsymmetrical draw. Therefore, sectional diagrams for each configuration are drawn, and based on estimated best recoveries with least dilution; optimum blast retreat distances are developed.

After all other parameters have been chosen, such as sub-level interval, pillar width, height and width of extraction drift and ring burden, etc. and the mine developmental work is started and kept ahead of production date
for a year or two. Blast retreat distance is, then, by far the most im¬
portant variable which may give any chance of flexibility in this method.

A change in blast retreat distance can occur by variation in moisture
content. This is a matter of concern for Granduc Mines because of small cover
of waste rock above the top most level. When the cave starts, it will pro-

Tably cover a large catchment area of a terrain which is covered with snow,
almost eight months in a year. Any hot air leaking through the cave might
accentuate this problem, unless some alternative means of water diversion are
devised. Also, varying fragmentation (due to the nature of ground, etc.) and
consolidation (caused by bigger or smaller explosive charges and/or by a var-
ible thickness of the over-lying mat of waste rock) hold different amounts
of moisture content. This type of study is useful from the point of view of
determining an optimum blast retreat distance.

There are two alternatives which can be considered from a practical
standpoint.

1. In the case of ring drilled sub-levels which will be kept in readi-
ness before actual production, generally with a uniform ring burden
throughout, the sub-level may have to use a blast retreat distance
which is an exact multiple of the ring burden in order to be most
profitable in particularly changed situation.

2. To study the variation in the depth of the ellipsoid or cave figure
beforehand with respect to particular rock type to be encountered in
the mine (rock may break between the limits of very fine to coarse)
and also expected wetness in the area - so that ring burdens may be
varied and designed according to the specific conditions for dif-
ferent portions of the same sub-level.
If the first alternative is more practical and acceptable, then there is a need of developing some flow corrective measures to cause the rock flow to conform with the nearest single or multiple ring burden—defined as blast retreat distance.

Principles of arching in the ideal soils have been utilized in order to know the change in the depth of the ellipsoid of motion behind the solid face.

From Terzaghi (40), arching is one of the most universal phenomenon encountered in soils both in the field and in the laboratory. If one part of the support of a mass of soil yields while the remainder stays in place, the soil adjoining the yielding part moves out of its original position between adjacent stationary masses of soil. The relative movement within the soil is opposed by a shearing resistance within the zone of contact between the yielding and the stationery masses. Since the shearing resistance tends to keep the yielding mass in its original position, it reduces the pressure on the yielding part of the support and increases the pressure on the adjoining stationary parts in commonly called 'arching effect' and the soil is said to arch over the yeilding part of the support.

Referring to Figure 4 the local yield of the horizontal support of a bed of granular material can be produced by gradually lowering a strip-shaped section \( \dot{a}.\dot{b}.\) of the support. According to the radial shear stress theory, as soon as the strip has yielded sufficiently in a downward direction, a shear failure occurs along two surfaces of sliding which rise from the outer boundaries of the strip to the surface of the granular material. In the vicinity of the surface, all the grains move vertically downward. This has been demonstrated by time exposure photographs, Terzaghi (40).
FIGURE 4. FAILURE IN COHESIONLESS GRANULAR MATERIAL PRECEDED BY ARCHING. (A) FAILURE CAUSED BY DOWNWARD MOVEMENT OF A LONG NARROW SECTION OF THE BASE OF A LAYER OF GRANULAR MATERIAL. (B) ENLARGED DETAIL OF DIAGRAM (A).
Such a movement is conceivable only if the surfaces of sliding intersect the horizontal surface of material at right angles. Hence, the surfaces of sliding have a shape similar to that indicated in Figure 4 by the lines a'c. and b'd. The equations for the surfaces of sliding a'c. and b'd. have not yet been solved. However, the experiments have suggested that the average slope angle of these surfaces decreases from almost 90° for low values of Z/2b to values approaching \((45° + \phi /2)\) for high values of Z/2b.

In the case of sub-level caving stopes, a high value of Z/2b exists near the draw point considering a fairly thick mat of waste rock overlying the blasted column of ore. Therefore, study of the variation in the angle \((45° + \phi /2)\) or the angle of sliding in the broken mass near the opening and the eventual development of the curved sliding surface is very important in determining the most suitable blast retreat distance.

Angle of internal friction, \(\phi\) used in the expression for angle of sliding \((45° + \phi /2)\), is a function of void ratio, moisture content, angularity, particle size and confining pressures, etc.

According to 'Mohr - Coulomb Failure Law'.

\[
\gamma = c + \sigma \tan \phi
\]

Where \(\sigma\) is the normal stress on the failure plane at failure and \(\gamma\) is the shear stress on the failure plane at failure, \(c\) stands for Cohesion and is equal to zero for Cohesionless materials and \(\phi\) is angle of internal friction.

Values of \(\phi\) in the preceding equation can be determined by means of laboratory tests.

In order to calculate the state of stress located immediately above the yielding strip - Terzaghi has assumed the surfaces of sliding are to be vertical as indicated by the lines a'e. and b'f. (Figure 4.). This way the
problem of computing the vertical pressure on the yielding strip becomes identical with the problem of computing the vertical pressure on the yielding bottom of the prismatic bins. The following formula is used to determine the vertical stress \( \sigma_v \) on a horizontal section at any depth \( Z \) below the surface for a Cohesionless material.

\[
\sigma_v = \frac{b \gamma}{K \tan \phi} \left( 1 - e^{-K \tan \phi \frac{Z}{b}} \right) \tag{3 (12)}
\]

Where \( K = \) Ratio between the horizontal and the vertical pressure - Taken as unity immediately above the yielding strip.

\( b = \) Half width of the strip (See Figure 5)

\( \phi = \) Angle of shearing resistance.

\( \gamma = \) Unit weight of the granular material.

Considering a unit width of the yielding strip and by substituting \( K = 1, b = 8, \phi = 42^\circ, \gamma = 100 \) lbs./cu. ft.

For \( Z = 60' \)

\( \sigma_v = 887.9 \) lbs./sq.ft. or 6.15 P.S.I.

For \( Z = 160' \)

\( \sigma_v = 887.9 \) lbs./sq. ft. or 6.15 P.S.I.

Corresponding normal stress (\( \sigma_h \)) on the vertical surface of sliding can be computed from:

\[
\sigma_h = K \cdot \sigma_v \tag{3 (13)}
\]

The above analysis is applicable to the flow in an open mass, but considering the particular case of a sub-level caving stope with side slopes - formulas derived for Convergent hoppers with surcharge (See Walker (10) ) for mass-flow are more relevant in calculating the nature of stresses close to the outlet in stopes.
The average vertical pressure \( \bar{V} \) is given by the following formula:

\[
\bar{V} = \frac{\gamma h_c}{C - 1} \left( 1 - \left( \frac{h_c}{h_o} \right)^C \right) + V_o \left( \frac{h_c}{h_o} \right)^C \tag{3.14}
\]

Substitute the following in Equation 3 (14)

\[
\gamma = 100 \text{ lbs./cu.ft.} \\
C = 3.13 \\
\text{(taken as the nearest approximation from Walker (10) Table 2; P. 994; for } \alpha = 15^\circ, \beta = 40^\circ \text{ and } \delta = 50^\circ \text{)}
\]

and from Figure 5

\[
\begin{align*}
h_c &= 41 \text{ feet} \\
h_o &= 60 \text{ feet} \\
V_o &= 100 \times 100 \text{ lbs./sq. ft. as the surcharge} \\
\bar{V} &= \frac{100 \times 41}{2.13} \left( 1 - \left( \frac{41}{60} \right)^2.13 \right) + 10,000 \left( \frac{41}{60} \right)^{3.13} \\
\bar{V} &= 28.5 \text{ P.S.I.}
\end{align*}
\]

A range of confining pressures derived from Equations 3 (12) and 3 (14) which represent the pressure at the outlets of different stope layouts coupled with a range of moisture content tried with varying sieve sizes of the material used were tested by means of Triaxial testing machine in the laboratory at U.B.C. Details of tests conducted and their results are presented in Chapters 4 and 5.

3.4 Design of Mass-Flow Bins v/s Stope Design:

In order to assess the sub-level caving stopes on the basis of mass-flow bin and funnel-flow bin condition, the following analysis shows that a variety of situations can develop. For example:

A. Transverse sub-level stopes: can have mass-flow condition on three
FIGURE 5. STANDARD TRANSVERSE SUB LEVEL CAVING LAYOUT.
sides and plug-flow condition on one side (back-end) up to the end of the side slopes to the next sub level above and beyond that height it has plug-flow condition on three sides and mass-flow condition on one side.

B. Longitudinal Sub Level Stopes:

1. Narrow & steeply* dipping stopes can have = 3 sides 1 side
2. Narrow & gently dipping stopes can have = 2 sides 2 sides
3. Wide ore bodies, steeply dipping stopes (2 Extraction drifts on same level) = 2 sides 2 sides
4. Wide ore bodies, gently dipping stopes (2 Extraction drifts on same level) = 1 side 3 sides
   F.W. drift
   or
   2 sides 2 sides
   H.W. drift

NOTE:
* Steeply dipping would mean: dip of 75° and above.

From the above it appears that a stope design will present many more variables as compared to the bin design and, therefore, Jenike and Walker theories of gravity flow as applied for mass flow in hoppers and bins need a considerable improvement before their application can be extended to the gravity flow of blasted rock in the stopes.
CHAPTER 4.

CONSTRUCTION AND OPERATION OF THE TESTING EQUIPMENT.

4.1  General:

The model and all accessories for it have been constructed to the scale of 1:30. Preliminary tests were made on 1:60 scale, but this scale was found to be too small to permit the drawing of sectional diagrams and allow proper study on the digging pattern. One test was made with 1:20 scale model and it was found much easier to work with, as the chances of disturbance and deviations are reduced.

4.2  Construction:

The model was made in the shape of a box, 3 ft. high x 4 ft. wide x 18 in. deep. (see Figure 6) It was made out of 2" x 4" lumber, with two vertical sides, moveable inside the box to allow different positions of the sub drifts, etc. The back wall was covered with 1/2 in. plywood, while the front wall was covered with 1/4" plexiglass. The frame construction was reinforced with 1/2" round bars, which strengthened the model. The plexiglass front was supported on the outside by horizontal irons to prevent bulging, etc. A provision was made for two side frames for the model, which permitted the extension of the model upwards as well as sideways, when desired. Upward extension is needed while doing multiple level tests and side extension is required for testing flat dipping longitudinal stopes. This was achieved by placing an additional plexiglass plate on top of the first one. This feature of building the model in portions provided ease of loading and unloading the model. The model box was placed on two low horses, so that scooping from its lowest part could be done comfortably.
FIGURE 6A. MINE MODEL ASSEMBLED FOR TRANSVERSE SUB LEVEL CAVING.
FIGURE 6B(ii) MINE MODEL ASSEMBLED FOR LONGITUDINAL SUB LEVEL CAVING.
4.3 **Ore and Waste Material:**

It was assumed that the maximum size of the ore from the production blasting should not exceed 20 in. and the average size in the range to 12 in. - 16 in. It is known, however, that coarse fragmentation makes the recovery worse. Average grade ore was brought from the mine and crushed in a small laboratory crusher. Choke feed into the crusher was avoided, since that produced more angular ore particles. With normal feed, the shape of the crushed rock down to a minimum size of 1/4 in., had approximately the same shape as that of the broken ore observed in some of the experimental drifts in the mine. Crushed rock was screened and then mixed in a proportion to have fragmentation distribution similar to what was expected in the actual operation. The screen analysis of the ore material used in each test is included in Table 4, of Appendix III. Approximately 5,000 lbs. of crushed ore material was prepared. The ore was then placed in the bins beside the model.

Although not much is known yet about the fragmentation of the caved hangingwall, it is assumed, for model testing, that size of waste is to be the same as that of the ore material particles and also, the gravity flow characteristics to be nearly the same as that of the ore. No ore and waste tests have been performed to determine the ore recovery and waste dilution estimates for reasons detailed in Section 5.4. However, for the purposes of demonstrational tests and for the loading intensity tests, etc., white dolomite crushed to the same size as that of ore was used in the model.

4.4 **Extraction Drifts and Loading Bucket, Etc.:**

The extraction drifts were made out of 1/6" galvanized plates, scaled down thirty times. The loading bucket was made of 1/16" steel plate
and its design is exact scaled down copy of the scooptram ST-4A. Its volume
is \((30)^3 = 27000\) times less than the scooptram bucket and its digging char-
acteristics are approximately the same. The bucket was pushed into the muck
pile, then turned around its pivot points by pressing down the digging handle
with the thumb at the same time the bucket was raised. One test was arranged
at 1:20 scale. For this purpose, a new set of extraction drifts and scoop-
tram bucket was made. The model box was the same as that of 1:30 scale tests.

Blast plates, used for separating the ore and waste columns during
loading of the model were made out of 1/16" sheet plate. (see Figure 7).

4.5 Operation of the Model:

The model tests were carried out in a future mill laboratory area
of the dimensions approx. 40 ft. x 20 ft. This place was available entirely
for model testing. It was quite convenient here to accommodate benches,
tables, screens and tools, etc., to do any fittings and carpentry work, etc.
Alterations to the model of some sort were required almost after every test
for the start of a new configuration.

Determinations of the cave figure or the draw figures have been
carried out with the model arranged as described above. A drift opening is
cut out at the bottom of the front plexiglass panel. A wooden block, with
the same shape as the drift hole, is pushed into the hole the same distance
as the desired blast retreat distance. This block was cut with an incline
in the front so as a natural angle of repose of the rock in the model could
be developed. The side slopes, in the case of transverse sub level caving
test, were placed on each side of the drift. Slopes were simulated by ply-
wood boxes with desired slope angle. For longitudinal sub level testing, the
FIGURE 7A  BLAST PLATES

FIGURE 7B  WOODEN BLASTING BLOCK, EXTRACTION DRIFT, AND BUCKET
the H.W. and F.W. panels were fixed at the desired angle and adjusted along the base of the model so as to adjust for the proper location of the drift with respect to the footwall. Whenever F.W. slash was needed to be studied, a false floor was built on the top of the F.W. panel so as a recess could be cut into it. See Figure 15 in Appendix V.

The model was loaded with the ore material by spreading 2 in. layers at a time and then placing marked stones on a special three dimensional pattern at each interface. Each such interface had a grid of 2 in. x 2 in. (5 ft. x 5ft., full scale). A total of 4,200 red coloured marked stones were required for the purpose. This system had its origin at the bottom of the model and 12 in. to the left of the drift. The origin was moved sometimes for flatly dipping longitudinal tests and this fact was noted on the particular test. The process of loading was repeated in this manner, and layer after layer was placed until the model was loaded high enough to provide a capping of about 20 inches, (50 ft., full scale) above the highest expected cave figure contour understudy. The blasting was simulated by pulling at the wooden block, whereafter, the loading could start. Five buckets were drawn at a time and marked stones were picked as they appeared in the tunnel. These stones were then plotted on sectional diagrams covering the entire depth of the model at 2 in. intervals (5 ft., full scale). Plotting was not done on the basis of cumulative weights because it is erroneous. Therefore, all plots were made on the basis of volumes drawn out of the broken mass. With the application of appropriate swell factors, these plots were modified. This point is explained further under Section 5.4.

After the requisite amount of scoops have been drawn from the model, it was unloaded and all the marked stones were recovered for reuse.
Ore material was properly mixed again, if required, and a swell factor test was made on the material before the next test was performed.

Side slopes made up of plywood boxes for testing transverse sublevel stopes and also the F.W. and H.W. panels meant to experiment longitudinal stopes, both were covered with sand paper to get the same friction coefficient as for the ore. Tests performed by Swedish experimenters have shown that the friction coefficients for rock on rock and rock on sand paper are almost equal to 0.7.

4.51 Tests With Vibrated Column of Ore:

A few check tests were made by vibrating the column of ore by means of a small hand vibrator as detailed in Section 5.4. The loading of the model was achieved by pushing a wooden block with the same shape as the drift hole into the hole the same distance as the desired blast retreat distance. A blast plate, which had the same shape as the blast, was put above the end of the drift block. Ore was filled to just above the block and, of course, behind the blast plate. The ore between the blast plate and the plexiglass was vibrated in vertical sections of 2 inches. Ore material was placed behind and in front of the blast plate to keep it in balance as the loading continued. When the ore had been vibrated up to the level of the blast plate then the blast plate was pulled out. From there on, the model was loaded normally in 2 in. layers and marked stones were placed on grid intersections until a capping of at least 20 inches was achieved. Thereafter, the procedure detailed under Section 4.5 was repeated and sectional diagrams were drawn.
4.52 Ore and Waste Tests:

Ore and waste tests were performed for the demonstrational purpose only. In a few tests, a 2 inch waste band of white dolomite was placed on the top of the contours of the desired shape of the blast. As the scoops were drawn from the bottom drift, the flow of this band was viewed from the front plexiglass panel to determine the proper loading sequence, i.e., either from the F.W. or H.W. side of the drift. Photos of the sequence of events were continuously taken. Two sets of these pictures are shown in Appendix V. Figure 14 shows a longitudinal stope with a single drift on two levels for a 30' orebody at 65° F.W. angle and Figure 15 shows longitudinal stope with two extraction drifts on the same level for a 50 ft. wide orebody at 55° F.W. angle. The cumulative number of scoops drawn either from the F.W. or H.W. side of the drift are shown on the sign placed in front of the model.

4.6 Triaxial Compression Testing Equipment:

These tests were performed to study the flowability of ore samples obtained from one particular place in the mine.

A number of testing machines are available in the soil testing techniques. For example, direct shear tester, ring shear tester and Triaxial shear tester, etc. A triaxial testing machine was used for the above tests, mainly because of its availability at the U.B.C. laboratories for the size of the sample tested. (A mixture of fines to 1/2" crushed rock.)

A cell for 4-in. diameter samples was used as the testing equipment. For compacted samples, this cell permits the use of standard compacting equipment. However, a vibrator was used to consolidate the sample. Samples
up to a maximum grain size of 3/4 inch may be tested, although satisfactory test specimens are more readily obtained if the limit is placed at the 3/8 inch sieve size, Bishop and Henkel (41). The height of the test specimen used was 8 inches.

Figure 8 shows the pictures of the test specimen and the testing equipment.

It can be argued that triaxial testing equipment used as above may have the following limitations.

1. Required degree of compaction and uniformity may not be obtained every time a test is repeated.

2. A new sample has to be formed each time. This factor is important because a large number of tests have to be performed for complete data. (In the case of Walker's Ring Shear Tester, any one sample may be used for a range of confining pressures, etc.)

3. Errors caused due to the use of rubber membrane around the sample. (Up to 0.6 P.S.I.)

4. Appropriate corrections need to be applied due to the weight of the sample itself - especially because of the low pressures existing near the outlet of the stopes. This aspect has been noted to be particularly important in Jenike's analysis, in which case, tall and narrow mass-flow bins with small outlet openings have pressures in the range of 1 P.S.I. near the outlet and this is one of the main reasons of his use of a low and small direct shear tester for design work.
FIGURE 8. TRIAXIAL TESTING EQUIPMENT

PICTURE 1. Sample ready for testing

PICTURE 2. Failed sample

PICTURE 3. Triaxial cell under loading (assembly)

PICTURE 4. Vacuum pump used for negative confining pressures.
However, in the case of stopes with comparatively larger opening size, this error may not be very serious, and it can be seen from the preceding Equations 3 (12) and 3 (14) that pressures are in the range of 6 P.S.I. to 30 P.S.I. depending on whether it is a free mass-flow, or flow on the side slopes, etc.

Observations made and results obtained from these tests are presented in Table 2 of Appendix III and in Chapter 5.
CHAPTER 5
TESTS DESCRIPTION PROCEDURES & RESULTS.

5.1 General:

The test work was started with the study of the cave figures in ore in the open mass. Cave figure determinations in open mass mean that the model was filled with ore material and marked stone and was without any obstructions against a free development of gravity flow and cave figures. In other words, no side slopes were placed on each side of the drift. Loading was done from the drift and sectional diagrams plotted as described in Section 4.5. These tests are useful in doing the solid geometrical studies of the different sequences of the flow and they are also of fundamental importance for the understanding of the gravity flow. One set of such cave figures tests is sufficient to study the transverse sub level caving method, but for the longitudinal sub level caving tests, such cave figures studies are essential almost with every possible configuration. This was observed during the preliminary model tests which were conducted by us before, and it was observed that the particle movements along the H.W. contact is faster than in the rest of the broken mass.

Therefore, first a few tests were performed in order to check the layouts of the transverse sub level caving which have been used in the planning work already. Thereafter, a few modifications were done with the model to render it suitable for the comprehensive testing of the longitudinal sub level caving layouts, this being the main area of investigation of this thesis. Instead of testing every possible configuration for the longitudinal sub level caving layouts, more emphasis has been given to check the design work of the 'A' ore body, which is generally narrow and steeply dipping.
The 'A' ore body is one of the first few areas on the mining program. After the procedures of testing and interpretation within a close range of accuracy have been established, extended test work could be taken up for any particular configuration at the time it will be required to check any layout.

5.2 Description of the Model Tests:

Transverse Sub-Level Caving Method: A total of eleven tests was done. These comprised of tests in the open mass and tests with side slopes of 74°, 79° and 84°. Therefore, the tests covered 5° above and below the standard adopted pattern of 79° side slopes. Two of the above tests were repeated in order to evaluate any discrepancies due to personnel error; these were found to be negligible. Tests performed in the open mass indicate that the angle of sliding is between 75° - 79°. It is realized, however, that this angle is a function of size distribution in the broken mass and other factors.

Longitudinal Sub-Level Caving Method: Twenty-six tests have been completed for this method. Ore body widths of 20', 30', 40' and 50', varying between footwall angles of 55° and 75° with increments of 5°, have been tried. For the first few tests, the extraction drift was placed in footwall contact and draw configurations were studied. Later, it was decided to place the drift well into the footwall where footwall angles were less than 70°. These results were compared with similar layouts which had used no footwall slash. Quite encouraging results were obtained with this improvement.

Based on the above considerations, the test program was evolved.
A total of thirty-seven tests, one including on 1:20 scale covered the entire programme. General breakdown of the tests is as follows:

5.3 Test Program:

A. TRANSVERSE SUB-LEVEL CAVING TESTS:

1. Cave figure tests in open mass.
2. Test with 74° side slopes - central loading.
   Test with 74° side slopes - alternate loading.
3. Test with 79° side slopes - alternate loading.
4. Test with 84° side slopes - alternate loading.

B. LONGITUDINAL SUB-LEVEL CAVING TESTS:

<table>
<thead>
<tr>
<th>Orebody Width</th>
<th>Footwall Angles</th>
<th>Footwall Slash</th>
</tr>
</thead>
<tbody>
<tr>
<td>20 ft.</td>
<td>60°, 65°, 65°</td>
<td>no slash</td>
</tr>
<tr>
<td></td>
<td>65°</td>
<td>90° slash</td>
</tr>
<tr>
<td></td>
<td>65°</td>
<td>85° slash</td>
</tr>
<tr>
<td></td>
<td>65°</td>
<td>80° slash</td>
</tr>
<tr>
<td></td>
<td>65°</td>
<td>75° slash</td>
</tr>
<tr>
<td>30 ft.</td>
<td>60°, 65°, 75°</td>
<td>no slash</td>
</tr>
<tr>
<td></td>
<td>65°</td>
<td>80° slash</td>
</tr>
<tr>
<td>40 ft.</td>
<td>55°, 60°, 65°</td>
<td>no slash</td>
</tr>
<tr>
<td></td>
<td>65°</td>
<td>80° slash</td>
</tr>
<tr>
<td>50 ft.</td>
<td>55°, 70°</td>
<td>80° slash</td>
</tr>
<tr>
<td></td>
<td>in drift 'A'</td>
<td></td>
</tr>
<tr>
<td>20 ft. (1:20 scale test)</td>
<td>65°</td>
<td>80° slash</td>
</tr>
</tbody>
</table>

Some of the tests mentioned in 'A' & 'B' above were repeated to check the effects of different ore material, swell factors, loading pattern,
and the effect of consolidating column of ore over a desired blast retreat distance, by means of a vibrator.

The tests were not carried out in any particular sequence or order, but were done after the elimination principle, i.e., information obtained from the previous tests was used for the next test in a progressive manner.

5.4 **Testing Procedure:**

After a few tests had already been completed, according to the description in Section 4.5, the particular aspect of loading the model was studied and tested, using existing facilities. The percentage of voids was tested both in the loosely packed and packed ore material, dependent upon the screen analysis of the material being used. Considerable consolidation could be achieved by packing by means of a vibrator. In order to study the effect of consolidation, the broken ore column for a particular blast retreat distance was vibrated. This was done on the assumption that consolidation takes place in the direction of the extraction drift, because the majority of the free face is in that direction. It was found that cave figures do change depending upon how the ore material was filled in the model and confirmed the view that gravity flow changes with the change in the bulk density (derived from the specific gravity and the voids in the bulk). This aspect has an important bearing on the calculation of the ore recoveries, dilutions and the total extractions based on the draw figures. From the foregoing, it became apparent that the use of proper swell factors for the choke blasted ore column and the muck in the bucket of a scooptram are important for the proper interpretation of the draw figures.

It is considered, therefore, that a model loaded with ore and waste
material without proper consolidations in the respective regions would serve no better purpose than being a visual aid only.

For the above reasoning, any quantitative analysis based on the draw figures obtained from the tests used as such for the calculation of percentage recoveries, waste dilutions and total extraction, etc., will be unrealistic and misleading. For the same reason, ore and waste tests in the model, which are also extremely time consuming from the sorting point of view, were dropped from the program. A few illustrative tests and loading intensity tests were done as mentioned in Section 4.3.

Draw figures are obtained, therefore, by means of section diagrams with the help of marker grid stones as detailed in Section 4.5, in the case of every configuration studied.

Testing was carried on with the vibrated column of ore (Section 4.51). After the completion of a few tests, it was found that the use of the vibrator is not controllable and the consolidations obtained are not consistent. Further, the consolidations obtained by this method still could not represent fully the conditions of the actual blasted column of ore, and corrections are needed to interpret the results. Therefore, these tests were not continued as a regular procedure.

The final procedure, however, evolved after the above experimentation and its description follows.

Loading of the model was done with ore material only, without any vibration. Marker stones were placed in the entire volume of the loaded mass, in each test. Scoops were drawn with the desired loading intensity (a ratio of muck drawn from the F.W. or the H.W. of the drift), which was sometimes guided by the waste band laid on the top of the outline of the
actual shape of the blast. Extraction volumes were corrected with the application of proper swell factors (Section 5.41) of the material in use. Calculations for ore recovery, waste dilutions and total extractions are made as considered in Section 5.43.

5.41 **Swell Factor:**

On reviewing the work done at Mt. Isa, (50), it is noted that a confined swell factor of 1.10 is used for the blasted column of ore, which is caused by the expansion when a ring is fired. When the ore is extracted, a further expansion occurs such that an overall swell factor is 1.5, but the intermediate swell factor between 1.10 and 1.5 is 1.37. In draw control, intermediate swell factor is an important variable.

In order to derive this swell factor of the ore material in the scoop bucket used for the model testing, a few experiments were conducted. The value found was not much different from the value of swell factor of the actual material used in the model in any test (see Table 1 & 1A, Appendix II). It can be said, then that in situ swell factor of 1.37 used for the heaped bucket in the mine is less than the overall swell factor of 1.5 because of consolidation brought about by the digging action of the scoop over a wide range of fragmentation in the draw point.

It should be mentioned here that the computed values of ore recoveries and total extractions are as good as the assumptions made in the calculations of the swell factor. With some preliminary experiments and field observations, swell factors have been derived for the broken rock loaded in the model and the in situ swell factor of the loaded bucket for calculation work. It is, however, known that there are not very precise
means available yet for finding the in situ swell factor of the granular materials. It has been observed by various workers that consolidated granular material dilates as flow ensues. Now that consolidation of the ore column region in the model is much less than that of the blasted column of ore in the stope and it was observed also in the model that draw proceeds faster in the upward direction in relation to the depth in the model and on this account alone, it can be predicted that waste dilutions and thereby total extractions show higher values in the model than what will actually happen in the mine.

5.42 Scoop Factor:

From the above, a scoop factor or the number of buckets to be drawn for a particular blast retreat distance has been calculated as follows:

\[
\text{SCOOP FACTOR} = \frac{\text{In-situ volume of rock blasted for the b.r.d. under investigation} \times \text{In-situ S.F. of the muck in the heaped bucket}}{\text{Volume of the heaped bucket (112 cu.ft.)}}
\]

For simplicity, scoop factors based on the above have been used purely for comparison purposes throughout the computation work. In fact, this figure will have to be adjusted in actual practice since the outline of the draw volume stays generally outside of the ore and waste inter-face in the stope.

5.43 Calculation Procedure for Ore Recovery and Total Extractions:

The volumes of ore and waste, which are contained by cave figures of different total extractions have been estimated for different sizes of
blasts. Since the cave figures are unsymmetrical, it is not possible to use volume integration and consequently no rotational solid could be established. Instead, the volumes of the cave figures were estimated by using prismoidal formulas, by which the areas of very thin slices of the solid figures are measured.

Figure 9 shows the procedure for the calculation of ore recovery, total extraction and waste dilutions, etc., in the special case, when the ore column has been vibrated and has a different S.F. than the rest of the mass. In a general case, i.e., when the S.F. of the entire mass is the same or \( x = y \) in Fig. 9, calculations are modified accordingly.

5.5 Test Results:

Ore recoveries, total extractions and waste dilutions are calculated as described in Section 5.43 for a range of sub level intervals and blast retreat distances. These are tabulated in Table 1 and Table 1A in Appendix II. It is not possible to discuss all the tests here but the most representative ones have been grouped appropriately in the above tables for comparison purposes. Table 1 and 1A, also include the intensity of loading in the extraction drift.

Hang-ups and disturbances were observed during scooping the muck from the model. However, cave figures are drawn up after the general pattern of draw which is represented in the proposed layout drawings.

Due to the probable inconsistencies in the determination of swell factor of the vibrated ore column in some of the tests, the calculated values of the results might show variance. For this reason, a few tests were repeated with swell factors determined as carefully as was possible
ORE PACKED WITH VIBRATOR

A TYPICAL SECTION ALONG THE CENTER OF THE EXTRACTION DRIFT

ORE RECOVERY $\% = \frac{\text{Volume } C \div x}{V} \times 100 = O_r$

TOTAL EXTRACTION $\% = \frac{(\text{Volume } C \div x) + (\text{Volume } D \div y)}{V} \times 100 = T_e$

WASTE DILUTION $\% = \frac{\text{Volume } D \div y}{(\text{Volume } C \div x) + (\text{Volume } D \div y)} \times 100 = W_d$

WHERE

$a =$ BLAST RETREAT DISTANCE UNDER STUDY

$b = x \cdot a$ (VIBRATED COLUMN OF ORE)

$V =$ IN SITU VOLUME OF THE ORE BLASTED IN THE BLAST RETREAT DISTANCE OF $'a'$

FIGURE 9. CALCULATION PROCEDURE FOR ORE RECOVERY, TOTAL EXTRACTION AND WASTE DILUTION.
within the practical range. Tests numbers 37, 36, 31, 33 and 32 for orebody widths of 20', 30', 40' at 65° F.W. angle and 50' at 55° and 70° F.W. angle respectively can be considered most representative for the calculated values of ore recovery, total extraction and waste dilution, etc.

From the study of the Table 1 & 1A (Appendix II), it can be said that the results obtained from geometrically scaled model has given useful information regarding the relative importance of various layouts or configurations, but there remains yet the problem of properly scaling-up the results to predict the behaviour of the stope with regard to the actual ore recoveries and total extractions.

It can be foreseen, that these figures will have to be constantly revised as more information is learned about the draw characteristics, both from the model tests, and from actual controlled draws in the sub level cave areas. A further modification of the draw volume will change total extractions according to the grade in the stoping area.

Before the discussion on tests is started, the following notations are described.

1. #, in the discussion on various tests refers to serial number in Table 1 or 1A in Appendix II.

2. Recommended layout parameters based on the analysis of the results are presented in Table V, Appendix IV.

3. Configurations 'A' to 'H' in Appendix IV show proposed layout patterns based on Table V, Appendix IV.

5.6 Discussion on the Longitudinal Sub Level Caving Tests.

5.61 20 ft. Orebody width.
5.611 At 60° F.W. angle @ 60 ft. S.L.I.

#1 at 7' b.r.d.; no F.W. slash; \( \theta_r = 59.5\% \), \( Te = 83\% \)
#2 at 7' b.r.d.; 80° F.W.slash; \( \theta_r = 85.0\% \), \( Te = 134\% \)

Comparing #1 & #2; \( \theta_r \) improves with F.W. slash. \( Te \) in #1 show less than even 100%.

This is erroroneous because the increased flow along the H.W. Contact removed all the marked stones much faster and later, waste was rilling in from the very top with no markers in it. So a part of the total extraction was not shown on the sectional diagrams and hence, calculated \( Te \) is less than 100%. This error can quite possibly appear with narrow orebodies and flatter angles. However, chances of this occurrence were removed by placing the marked stones much beyond the top outline of the blast.

5.6121 At 65° F.W. angle, 60 ft. S.L.I. @ 6 ft. b.r.d.

#3, no F.W. slash ; \( \theta_r = 71.0\% \), \( Te = 130\% \)
#4, 90° F.W. slash ; \( \theta_r = 86.6\% \), \( Te = 146\% \)
#13, 85° F.W. slash ; \( \theta_r = 84.6\% \), \( Te = 138\% \)
#6, 80° F.W. slash ; \( \theta_r = 82.6\% \), \( Te = 128\% \)

Comparing the above, it shows that 90° F.W. slash gives the best ore recoveries, but specifications of production drill jumbos suit best to the F.W. slash of 80° to the horizontal, while keeping a minimum portion of the drift width into the waste rock. Therefore, layouts with 80° F.W. slash (#6) can be selected even though it produces slightly lower Ore recovery compared to 90° or 85° F.W. slash.

It should be mentioned here that total extractions used for any comparison of ore recovery are based on an assumed average grade of the.
orebody. However, total extractions would have to be adjusted according to high or low grade ore in any particular portion of the orebody.

Now, comparing Test No. 27 (#5, 6) and Test No. 37 (#19, 20, 21, 22, 23, 24) at 60 ft. S.L.I. and 80° F.W. slash, it shows that best ore recoveries and total extractions are obtained on a comparative basis, at 8 ft. b.r.d. Therefore, #20 would dictate the selection of optimum layout parameters at 60 ft. S.L.I.

5.6122 At 65° F.W. angle @ 45 ft. S.L.I.
   #15 at 7' b.r.d. @ 85° F.W. slash
   #26 at 6' b.r.d. @ 80° F.W. slash

give good ore recoveries and they are comparable to #20 as discussed in Section 5.6121. Therefore, 45 ft. S.L.I. can be a possible layout parameter but a 60' S.L.I. is more economical in the similiar conditions. Further, the selection of S.L.I. is to satisfy the restraint of either 30 ft. or multiple of it in the case of Granduc overall development program in the currently developing ore block.

5.6123 At 65° F.W. angle & 30 ft. S.L.I.

Test No. 28 (#17) and Test No. 37 (#27, 28, 29) show low ore recoveries compared with 45 ft. or 60 ft. S.L.I. in their respective tests.

It is easy to interpret that for the same percentage ore recovery and total extractions, a higher sub level interval which is compatible with other practical & operational considerations is more economical and, therefore, preferable.

From Sections 5.611, 5.6121, 5.6122 and 5.6123, #20 in Table 1 is
the selected layout. It is shown as Configuration A and Serial No. 1 (Table 5) both grouped under Appendix IV.

5.62 30 Ft. Orebody Width:-

5.621 At 60° F.W. angle, 60 ft. S.L.I. & Loading Intensity
   = H.W.: F.W.: 1:3
   Test No. 9 (#34, 35, 36, 37, 38 and 39), with no F.W. slash, shows on a comparative basis that 60° S.L.I. and 8 ft. b.r.d. is the best. However, it was found with further testing that overall ore recovery values can be improved with the introduction of a F.W. slash.

5.622 At 65° F.W. angle, 60 ft. S.L.I., no F.W. slash & 8 ft. b.r.d.
   #41, loading intensity = H.W.:F.W.:3:1 ; \( \theta = 54.7\% \) \( Te = 110\% \)
   #40, loading intensity = H.W.:F.W.:1:1 ; \( \theta = 81.0\% \) \( Te = 113\% \)
   The above shows clearly that alternate loading is better than excessive loading from the H.W. side.

   A comparison of draw figures representing #41 and #40 is shown by Configuration J & K (Appendix IV) respectively. A further improvement of the draw figure is brought about by a F.W. slash and an increased intensity of loading on the F.W. side at the start of loading. This is shown by Configuration B (#49, Section 5.6221).

5.6221 At 65° angle, 60 ft. S.L.I. & 80° F.W. slash.
   Loading Intensity = First 90 scoops are drawn from the F.W. side only and thereafter, alternate loading is done from each side of the drift.
Test No. 36 (#47 to #52) shows that 8 ft. b.r.d. (#49) renders the best ore recovery and least total extraction with the above layout parameters.

5.6222 At 65° F.W. angle, 45 ft. S.L.I. & 80° F.W. slash.
Test No. 36 (#53, 54 and 55) show ore recovery is better with 6 ft. b.r.d. compared to either a 5 ft. or 8 ft. b.r.d.

5.6223 At 65° F.W. angle, 30 ft. S.L.I. & 80° F.W. slash.
Again, Test No. 36 (#56, 57 and 58) show that b.r.d. of 6 ft. is better than 5 ft. or 8 ft. But on the whole, 30 ft. S.L.I. gives low ore recoveries and hence it is not preferable.

From Sections 5.622, 5.6221, 5.6222 and 5.6223, it is seen that #49 of Test No. 36 suggests the best layout pattern, which is shown as Configuration B and Serial No. 3 (Table 5) in Appendix IV.

5.623 At 75° F.W. angle & no F.W. slash.
Test No. 25 (#59, 60, 61, 62, 63 and 64) indicate that out of all 'sub level intervals' and 'blast retreat distance' combinations, #60 is preferable. Therefore, 60' S.L.I. at 8' b.r.d. is acceptable. Proposed layout is shown as Configuration F and Serial No. 4 in Table 5, both in Appendix IV.

5.63 40 ft. Orebody Width:-

5.631 At 55° F.W. angle, No F.W. slash and Loading Sequence
= H.W.:F.W.:::1:3
Test No. 14 (#65 to #70) indicate that 30 ft. S.L.I. is the best compared to 45 ft. or 60 ft. S.L.I. at b.r.d. of 5 ft. or 6 ft.

It is clear from this test that if no F.W. slash is used, then for
moderately wider orebodies (30 ft. to 50 ft.) at low F.W. angles (55° and below) a lower sub level interval with a small b.r.d. is desirable. Nevertheless, with the introduction of proper F.W. slash, ore recoveries can be improved further.

5.532 At 65° F.W. angle, no F.W. slash and Loading Sequence

= H.W.:F.W.::1:3

Test No. 13 (#71 to #76) shows that 45 ft. S.L.I. at 8 ft. to 7 ft. b.r.d. gives better recoveries compared to either of 30 ft. or 60 ft. S.L.I.

It is interesting to note here that with increase in F.W. angle to 65° from 55° as from Section 5.631, optimum S.L.I. increases to 45 ft. in place of 30 ft. However, this S.L.I. cannot be isolated from the rest of the operation, so the obvious selection would be in favour of a 60 ft. S.L.I. at 5 ft. to 6 ft. b.r.d. even though 30 ft. S.L.I. or 45 ft. S.L.I. give equal or better ore recoveries.

5.631 At 65° F.W. angle, 80° F.W. slash & Loading Intensity = First 180 scoops are drawn from the F.W. side only and, thereafter, alternate loading is done from each side of the drift.

Test No. 31 (#77, 78, 79, 80, 81, 82, 83 and 84) compared 60 ft., 45 ft. and 30 ft. S.L. Intervals over a range of blast retreat distances. 60 ft. S.L.I. with 8 ft. b.r.d. (#78) gives the maximum optimum ore recoveries. Therefore, the final layout is suggested by this. Configuration C from Serial No. 5 of Table 5 in Appendix IV is the proposed layout.

A graph showing the comparison between ore recoveries v/s total extractions for the recommended layouts of 20', 30' and 40' orebodies is attached in Figure 11.
FIGURE II. COMPARISON OF ORE RECOVERIES Vs TOTAL EXTRACTIONS FOR 20', 30' AND 40' OREBODIES AT 60' SUB LEVEL INTERVAL, 65° F.W. ANGLE AND 8' BLAST RETREAT DISTANCE
5.64  50 ft. Orebody Width:

5.641 At 55° F.W. angle, F.W. slash in Drift A (see Configuration D) and Loading Intensity (see Table 6 in Appendix IV).

Test No. 33 (#85, 86, 87 and 88). Calculated ore recoveries and total extractions are determined for 8 ft. and 6 ft. b.r.d. separately for loading in Drift A. Similar calculations are made for b.r.d. of 6 ft. and 4 ft. in the case of Drift B. Thereafter, various combinations of blast retreat distances in Drift A and Drift B are tried (#93, 94, 95 and 96). It is seen that a combination of 6 ft. b.r.d. in Drift A and also a 6 ft. b.r.d. in Drift B (#95) result in better ore recoveries compared to other combinations. Hence, this combination is acceptable for layout in this case. Loading sequence adopted for Drift A and Drift B, given in Table 6 - Appendix IV, was guided by the progressive downward movement of a band of waste rock (white in colour) placed on the top outline of a standard ring at the start of loading. Pictures were taken at different stages of loading and they are grouped under Figure 15 in Appendix V. Proposed layout is shown as Configuration D from Serial No. 6 of Table 5.

5.642 At 70° F.W. angle, F.W. slash in Drift A (see Configuration E) and Loading Intensity (see Table 7 in Appendix IV).

Test No. 32 (#89, 90, 91 and 92). Exactly the same procedure is adopted as in the case of Test No. 33, Section 5.641. Calculated ore recoveries and total extractions are determined for 8 ft. and 6 ft. b.r.d. for Drift A and 6 ft., 4 ft. b.r.d. for Drift B respectively. From different combinations of blast retreat distances in Drift A and Drift B (#97, 98, 99 and 100), an 8 ft. - 6 ft. combination (#98) gives the highest overall ore recoveries. Proposed layout is based on this result and is shown as Con-
figuration E from Serial No. 7 of Table 5.

In the selected layout, combination of blast retreat distances cover different depths in Drift A and Drift B. Therefore, proper sequence of blasting would have to be developed in actual practice. In any case, if different blast retreat distances are chosen for Drift A and Drift B, then retreat in Drift B should be kept a step ahead of retreat in Drift A. The foregoing is specially significant with orebodies dipping at angles less than 65°. Loading sequence from Drift A or Drift B is shown in Table 7, Appendix IV.

In order to find the most suitable location for Drift 'B' in relation to Drift A, the first design was made by transposing different cave figures already obtained on the sections of the wider orebody, so as to obtain the best draw. As the testing progressed, a noteworthy observation was made in the development of a vertical plane of draw which results from the extension of surfaces of sliding (Section 3.37) starting from the inner sides of the extraction drifts. Therefore, it is found that optimum maximum centre to centre distance between Drift A and Drift B is double the horizontal distance between the centre line of either drift from the vertical position of the surface of sliding. This phenomenon is of considerable value and can be used in the quantitative design of locating multiple extraction drifts in the case of longitudinal sub level caving methods thus eliminating the guess work employed so far. A logical extension of this procedure could be applied to 60 ft. or even bigger orebody widths.

5.65 Estimated Waste From the Footwall Slash:-

Footwall slash to be taken in the waste rock for the purpose of improving the gravity flow in longitudinal sub level caving stopes is
recorded below as a percentage of the total volume of ore broken in a standard ring:

<table>
<thead>
<tr>
<th>Orebody Width</th>
<th>F.W. Angle</th>
<th>Extra Waste in the Slash</th>
</tr>
</thead>
<tbody>
<tr>
<td>20'</td>
<td>55°</td>
<td>13.7%</td>
</tr>
<tr>
<td>20'</td>
<td>65°</td>
<td>7.8%</td>
</tr>
<tr>
<td>30'</td>
<td>55°</td>
<td>9.0%</td>
</tr>
<tr>
<td>30'</td>
<td>65°</td>
<td>5.0%</td>
</tr>
<tr>
<td>40'</td>
<td>55°</td>
<td>6.7%</td>
</tr>
<tr>
<td>40'</td>
<td>65°</td>
<td>3.7%</td>
</tr>
<tr>
<td>50'</td>
<td>55°</td>
<td>2.8%</td>
</tr>
<tr>
<td>50'</td>
<td>70°</td>
<td>1.9%</td>
</tr>
</tbody>
</table>


Towards the end of the test work, one test was conducted on a 1:20 scale geometrically reduced model with similar basic parameters as used in Test No. 37 at 1:30 scale. Also, the parameters of testing were kept similar in both cases. The swell factor of the material used in this test was arranged to match with the equivalent test (No. 37) on 1:30 scale. This was achieved by changing the screen analysis of the tested material in steps until the desired swell factor was obtained. From Test 35 (#30, 31, 32, 33), 60 S.L.I. with 8 ft. b.r.d. gives the highest ore recoveries amongst this group. From this it is clear that results obtained from 1:20 scale and 1:30 scale show a good agreement with each other.

Barnes (39), in his paper "Similitude in the Studies of Tillage Implements Forces", has used this technique of checking the predictions made by small scaled models with a bigger model than before and repeated this procedure a reasonable number of times. This exercise helped show the important variables taking part in the process and, as a result, close
predictions to the prototype behaviour were made. Similar approach can be adopted in the mine model work to further advantage.

5.7 Discussion on the Transverse Sub Level Caving Tests:

5.71 At 79° side slope & 30 ft. S.L.I.
Test No. 4 (#101 to #105) show that blast retreat distance in the range of 7 ft. and 6 ft. (#103 & #104) give best ore recovery values.

5.72 At 74° side slope & 30 ft. S.L.I.
From test No. 6 (#106 to #109) is found that in this case also, best recoveries are to be found with blast retreat distances of 6 ft. to 7 ft.

5.73 At 84° side slopes & 30 ft. S.L.I.
Test No. 11 (#114 & 115), here also 6 ft. and 7 ft. blast retreat distances give high ore recoveries.

It is clear, however, that optimum side slope angle must be found somewhat above the angle of sliding, which was found to be between 74° and 79° as determined from the cave figure tests in the open mass. From Sections 5.71, 5.72 and 5.73 above, ore recoveries show up better with 84° side slopes as compared to either 74° or 79° side slope. However, lower development cost and operational costs required with a layout pattern of 79° side slopes, as compared to 84° side slopes, make it a desirable choice even though somewhat better ore recoveries are obtainable with 84° side slope. A 30 ft. S.L.I. and 79° side slope pattern has already been used in the planning work. This is shown by Configuration H from Serial No.8 of Table 5. Configuration I shows the pattern of draw in a vertical section along the centre line of the extraction drift.
In conclusion, it can be said that the design of these proposed layouts permit adoption to the already existing development work, when flexibility is of great value for the first period of operation.

5.8 Change in the Angle of Sliding - Determined by Triaxial Compression Testing Equipment:

Table 2, attached in Appendix III - records some of the tests performed, which clearly show the change in the angle of sliding and thereby a change in cave figures upon varying the test conditions. These tests were carried out with the ore samples from one particular place in the mine, but samples of ore collected from various locations in the mine representing different composition, can be tested to determine the difference in the flow properties for the purposes of quantitative design work.

To illustrate the importance of Triaxial tests in designing the blast retreat distance, attention is drawn to Test Nos. 10, 11, 12, 13 and 14 of Table 2, Appendix III.

<table>
<thead>
<tr>
<th>Test No.</th>
<th>Moisture Content (M.C. %)</th>
<th>Angle of Sliding (Degrees)</th>
</tr>
</thead>
<tbody>
<tr>
<td>10</td>
<td>NIL</td>
<td>65°</td>
</tr>
<tr>
<td>11</td>
<td>1.8%</td>
<td>64°</td>
</tr>
<tr>
<td>12</td>
<td>3.6%</td>
<td>69°</td>
</tr>
<tr>
<td>13</td>
<td>7.2%</td>
<td>61°</td>
</tr>
<tr>
<td>14</td>
<td>10.8%</td>
<td>51°</td>
</tr>
</tbody>
</table>

Figure 10 shows that the angle of sliding for the dry sample (Test No. 10) is 65°. A moisture content of 1.8% (Test No. 11) does alter the angle of sliding to 64°, but it is not very significant.
ANGULAR OF SLIDING WITH VARYING MOISTURE CONTENT
DETERMINED BY TRIAXIAL TEST
Moisture content of 3.6% (Test No. 12) reduces the flowability of the mass and tends to steepen the angle of sliding to 69° - hence reduces the depth of ellipsoid.

Whereas, moisture content of 7.2% and 10.0% (Test Nos. 13 and 14) show a progressive increase in the flowability, and flatten the angle of sliding to 61° and 51° respectively - hence increase the depth of the ellipsoid of motion.

Material description "F" (Table 3, Appendix III) was used in tests 10 through 14, for the above comparison. It is noted that Jenike (38) used an arbitrarily chosen No. 20 mesh material in his tests, because the flowability of a solid containing a range of sieve sizes, which includes both fine and coarse particles, is invariably governed by the flow properties of the fine fraction. This is explained by the fact that during flow the shearing takes place across the fines.

Nevertheless, these tests do indicate the importance of extended study in this direction, with regard to Granduc ore and waste.
CHAPTER 6

CONCLUSIONS

While full scale testing under natural conditions would remain essential as final demonstration of the worth of these kind of tests, certainly much preliminary screening of the trial layouts of Longitudinal Sub Level Caving Method could be conducted on the model.

Qualitative observations are reported and the effects that will have to be dealt with in theoretical treatment are described. The following conclusions are based on the results of a few most representative configurations of orebodies at Granduc Mines, and hence, extrapolation of some of these conclusions should be avoided until further data on all possible configurations is available.

1. From the theory of models, analysis completed at the present stage shows that geometrical reduction of the model is permissible, so that ore recovery and total extraction figures obtained from the model tests may at least be used usefully for comparison purposes. However, a correction for the conversion of results based mainly on the consolidation and bulk density in the stope is required.

2. Tests conducted on 20, 30, 40 and 50 ft. orebody widths at footwall angles less than 70°, with a footwall slash, which is compatible with production drill Jumbos specifications have shown improved recoveries at 60 ft. sub level intervals.

3. Higher sub level intervals of 60' with either narrow and steep orebodies or narrow and medium dipping orebodies with proper F.W.
slash show better recoveries compared to a lower sub level interval of 30' or 45'. However, wide and flat dipping orebodies are more suited to lower sub level interval of the order of 30 ft.

4. It has been observed, with the aid of triaxial testing equipment, that there is a change in the cave figures when the confining pressures or the moisture contents are varied. Study of the change in angle of sliding can, therefore, be useful in the design of optimum blast retreat distance, etc., on the quantitative basis.

5. Tests on 50 ft. orebody width, with two extraction drifts on the same level on a longitudinal sub level caving method, has revealed that overall ore recoveries are low at low footwall angles, but with steeper footwall angles (70° and above), acceptable ore recoveries are obtainable. A layout with twin drifts at 60 ft. sub level interval with longitudinal sub level caving is more economical in terms of waste development footage per ton of ore mined and also, overall ore recovery at steeper dips compared to transverse sub level caving method at 30 ft. sub level interval. Further, it is seen from the draw figures for 50 ft. orebodies that this method can be extended to 60 ft. orebodies also, with proper control of loading intensity in drifts 'A' and 'B'. This is where there is much to gain primarily from the development cost standpoint by the proper selection between the longitudinal or the transverse sub level caving method in a particular situation.
CHAPTER 7

RECOMMENDATION FOR FURTHER WORK AND DISCUSSION.

7.1 Recommendations for Future Work:

The literature review contained in this report, while not all-inclusive, reveals that the present state of understanding in the field of gravity flow as applied to the stope design is very limited. It may, therefore, be concluded that this field has quite a challenge in the development of modern sub level caving methods.

The results in the Table 1, Appendix II, are not conclusive, primarily because of several assumptions made in the calculations of ore recoveries and waste dilutions, etc. However, these results are useful in determining the relative importance of various layouts.

It is appropriate to make recommendations for other investigations, in the light of what has been revealed from the work so far. With this in mind, the following such recommendations for further and advanced work are offered.

1. Stoping layouts should be designed on a quantitative basis and use can be made of Jenike and Johnson and Walker theories and also their testing equipment in determining the flowability of the broken ore and waste. Based on such flowability test data, a mine model can be constructed to check the predictions before implementing the design in the mine.

2. Further experimental work could be developed to introduce the effect of blasting in the model, or some scaled effect thereof.
This would mean that analysis of the model scaling laws has to be improved upon so that model predictions are at least variance with reality.

3. It is seen that mine model test procedure adopted so far has been time consuming because of the placement and recovery of specially numbered rocks in the model on three dimensional grid. The numbered rocks were used as indicators in order to draw sectional diagrams of the cave figures. There is a scope for improvement here in increasing the efficiency of the testing procedure.

4. Observations in the field shall have to be made to check the merit of any design adopted. This, generally, is an extremely slow process since each problem usually has features peculiar to itself. However, techniques and instrumentation should be developed.

(a.) To monitor the flow pattern in the stopes so as any corrective measures be taken based on measurement rather than on hunch (monitoring in the mass-flow bins can be easily installed and has been used with some success, but for the plug-flow type bins proper monitoring has yet to be developed).

(b.) To determine the extent of caving in the stoping area which is important from the operational as well as the safety point of view, i.e., in order to know if there is an adequate mat of waste rock on the top of a particular column of ore at any time and also in assessing a situation beforehand when dangers from an air blast are imminent.
(c.) For flow promotion and flow correction in the stope on en­
countering an area which reacts exceptionally unfavourably
to the average conditions for which the layout was designed.

5. Only vertical blast/drill fans have been tested in the model so far. However, a few tests should be done with inclined fans, as this may be the best proposal just in case the fragmentation relationship is such that the caved waste is finer than the blasted ore in the mine.

6. The model tests could not give exact answers to the question of optimum layout, but do provide a very close range for it. The lay­
outs which have been proposed provide for adjustments in some parts. For example, blast retreat distance, inclination of fans and the intensity of loading, etc., can be improved according to the opera­
tional experience. Therefore, a practical follow-up should be taken up in the mine in a selected test area. This test area should be so chosen that it does not depend on the regular produc­
tion operation. Drilling, blasting and loading must be carried out with attention and, therefore, more slowly than normal so a good control can be kept by the person in charge of this operation.

7.2 **Operational & Practical Detail:**

Some of the parameters of sub level caving method are discussed here in the light of operating and practical detail.

7.21 **Fragmentation:**

It is known that fragmentation of ore and waste is the key to other parameters. How much fine ore should be broken, depends on the appearance
of waste from the cave. As the ore is costly to break it would, of course, be desirable to accept a rather coarse fragmentation. However, one should not look too seriously at the blasting costs, but permit a fragmentation which will be most profitable for the entire operation.

7.22 **Sub Level Interval:**

One can find from the model tests, the best possible sub level interval with normal conditions. But sometimes much attention has to be given to the irregular shape of the orebody as well as to its dip and pitch on making such a decision.

Another factor in this regard is the ease of blasting and fragmentation of ore. Higher sub level intervals require longer drill holes which increases the drilling costs rapidly, especially if closer hole placing has to be used to achieve acceptable fragmentation. Thus, there is an economical and also practical limit for each type of ore, which must not be exceeded.

7.23 **Fan Inclination:**

Optimum fan inclination is determined which is compatible with ore recovery, longhole drilling efficiency, safety and other operational problems.

Theoretically, 90° rings give an overall recovery which is better compared with the inclined rings. Therefore, this inclination should be used in the first layouts. However, there seem to be other reasons for hole fans inclined forward 10° - 20°.

Vertical rings have certain potential operational disadvantages from the drilling point of view; whereas, inclined holes have the advantages as the collaring is done further away from the operator and, therefore,
water and sludge do not fall directly onto the machine or the operator. It is also easier to charge the drill fan closest to the front if the holes are inclined forward as more room will be available between the back and the muck pile. This, also, creates a stronger brow which is most important to have in bad grounds.

With the H.W. retreat method (i.e., retreating from F.W. to H.W.), drill footages are 20 - 25% less with inclined holes compared to vertical angles with orebody dips in the range of 65°. This point should be strongly considered in the case of Granduc Mines where the H.W. retreat method has been planned.

7.24 Blast Retreat Distance:

Blast retreat distance or the blast depth determined from the model tests are only valid for vertical hole fans. If inclined hole fans are to be used, the blast depth will be smaller. However, as soon as results from the follow-up system (Section 7.1) are obtained, the blast retreat distance should be adjusted if necessary.

7.25 Time Factor:

It is known that time has certain effects on the flow properties of granular materials. If the blasted column of ore is left unloaded for some time, the solids remain under action of static pressure and, therefore, consolidated, which reduces their flowability. Further, if water is present, the cohesion forces in the blasted ore still strengthen it, at which serious hangups or doming can occur. Therefore, the blasts should not be left unloaded too long. Field investigations will be necessary to study this effect in full.
7.26 The following additional points may be considered with a view to practical trial:

A. It has been found necessary, at some mines, to reduce the fan burden to five feet and the toe burdens to 4½ - 5 ft. depending on how "tight" the position is. If bigger burdens are tried, it would probably be necessary to drill easier holes as shown in Figure 12.

B. One problem that arises from drilling all of a stope from an ore extraction drift is that there is a tendency to get "tight" corners, or in other words, a successive narrowing of break. In a normal longitudinal sub level caving layout (Figure 12), the contact between the H.W. and the fans tends to be tight. On the F.W. side, this corner is not so tight because it is normal to drill along the contact thus ensuring a break.

It may be possible to consider re-orientating the fans (in plan) as shown in Figure 13, to ease slightly the breaking of the ring. Alternatively, but perhaps less desirably from the point of view of recovery, the fans could be drilled in a vee-shaped normal to the footwall, but at an obtuse angle to the hangingwall.

7.27 **Intensity of Loading:**

Loading/mucking intensity is expressed as a proper ratio of loading either from the F.W. or H.W. side of the extraction drift. This is extremely important in the case of longitudinal sub level caving method. To ensure proper functioning at the operating level, it would require operator training, closer supervision and introduction of a bonus system based on the quality of much produced rather than on the total tons of rock removed from the stope.
THE HOLES FOR THE EASER RING (BROKEN LINES) SHOULD BE DRILLED FROM THE SAME SET-UP AND ANGLED FORWARD SO THAT THE COLLARS ARE NOT DAMAGED BY THE PREVIOUS BLAST.

VERTICAL SECTION ALONG THE LONGITUDINAL AXIS OF EXTRACTION DRIFT

SECTION I-I

FIGURE 12. LONGITUDINAL SUB LEVEL CAVING—POSSIBLE POSITIONING OF EASER HOLES.
ON NORMAL LONGITUDINAL SUB LEVEL CAVING LAYOUTS
THIS CORNER IS TIGHT

THIS CONTACT IS NOT SO TIGHT AS THE H/W BECAUSE IT IS POSSIBLE TO DRILL UP THE CONTACT.

SECTION

FOOTWALL DRIVE

PLAN

FIGURE 13. LONGITUDINAL SUB LEVEL CAVING— VEE SHAPED LONG HOLE FANS.
7.3 Comments on the Quantitative Design of the Stopes:

Quantitative design of bin is easier to achieve than the design of stope layouts for the following reasons:

1. The slopes of the hopper portion of the bin can be designed according to the flow properties of material it is charged with. Now, in the case of stope design, flow properties of the ore and waste rock can be determined and side slopes for the Tr. sub level caving stopes it may be possible to achieve the designed slope on the F.W. side only by slashing (when the designed slope called for is steeper than the dip of the orebody) as long as the slash is economical overall, whereas the maintenance of steeper slopes on the H.W. side may not be possible mainly because of the prohibitive length of longholes to be drilled from the level below specially on a sub level interval of 60' or higher.

2. Consolidation of material which occurs in the bins is either caused by the dead loads or due to the falling material on the top of the bin. These consolidations can be simulated on the tested samples for flowability tests. Whereas, the consolidating effect of blasting may not be completely reproducible on the tested sample meant to determine the flow properties of ore and waste in the stope.

3. In the quantitative design of bins, generally the storage and flow of only one type of bulk solid is considered whose flow properties are determined before hand. In the case of a stope, especially where broken ore and caved waste rock have different flow properties, continuous intermixing of ore and waste in the ellipsoid of motion
may produce products of variable flowability. The design of modern sub level stoping methods would be considered fully successful only when problems arising from a specially changed situation of flow caused by fragmentation or moisture content, etc., in a layout designed for average conditions could be dealt with effectively so as to ensure the extraction of planned ore recoveries and waste dilutions. There is a vast scope for improvement which could be brought in the design of sub level caving methods and its importance can easily be realized - say for example, even if a 5% increase in the overall ore recoveries could be achieved, then for Granduc Mines alone this could mean an additional income of approximately $29.0 million. (5% of approximately 36.0 million tons of ore expected to be mined with sub level caving methods @ say $16.0 ore).
LIST OF REFERENCES


APPENDIX I

The pertinent variables considered in the analysis of the gravity flow in the stope, are as follows:

<table>
<thead>
<tr>
<th>VARIABLE:</th>
<th>NOTATION</th>
<th>DIMENSION</th>
</tr>
</thead>
<tbody>
<tr>
<td>1. Draw Volume</td>
<td>V</td>
<td>L^3</td>
</tr>
<tr>
<td>2. A significant distance</td>
<td>l</td>
<td>L</td>
</tr>
<tr>
<td>3. Any pertinent distance</td>
<td>(\lambda_i)</td>
<td>L</td>
</tr>
<tr>
<td>4. Area of the opening (free flow)</td>
<td>A</td>
<td>L^2</td>
</tr>
<tr>
<td>5. Hydraulic diameter of the opening</td>
<td>(D_h)</td>
<td>L</td>
</tr>
<tr>
<td>= 4 x free area - Perimeter</td>
<td></td>
<td></td>
</tr>
<tr>
<td>6. Specific weight of the material (dead load)</td>
<td>w</td>
<td>ML^{-2}T^{-2}</td>
</tr>
<tr>
<td>7. Average size of the particle</td>
<td>d</td>
<td>L</td>
</tr>
<tr>
<td>8. Head of packing above opening</td>
<td>H</td>
<td>L</td>
</tr>
<tr>
<td>9. True density of solids</td>
<td>(\rho_s)</td>
<td>ML^{-3}</td>
</tr>
<tr>
<td>10. Bulk density of packing</td>
<td>(\rho_b)</td>
<td>ML^{-3}</td>
</tr>
<tr>
<td>11. Volume of the container</td>
<td>(V_c)</td>
<td>L^3</td>
</tr>
<tr>
<td>12. Digging depth of the scoop</td>
<td>(d_d)</td>
<td>L</td>
</tr>
<tr>
<td>13. Velocity of discharge from opening</td>
<td>v</td>
<td>LT^{-1}</td>
</tr>
<tr>
<td>14. Angle of internal friction</td>
<td>(\theta)</td>
<td>-</td>
</tr>
<tr>
<td>15. Angle of side slopes (angle of inclination of the hopper bottom)</td>
<td>(\theta)</td>
<td>-</td>
</tr>
</tbody>
</table>

So the volume of draw coming out of the stope is a function of:

\[ V = f (l, \lambda_i, A, DL, W, d, H, \rho_s, \rho_b, V_c, d_d, v, \theta, \theta) \]  \( - (1) \)

It is to be noted that the following simplifying assumptions have been made before proceeding with the analysis:
1. No effect of moisture has been considered.
2. No other forces than of the mass of broken rock above or on the sides of the stopes has been considered.
3. $\rho_b$ of the ore and waste is assumed to be the same in the area of draw.
4. Consolidating pressures offered by the blasting action has not been considered.
5. Generally equal loading of the material has been done from either end of the opening for Transverse Sub Level Caving Method.

From the theory of models, if the general equation for the prototype is:

$$\Pi_1 = F(\Pi_2, \Pi_3, \Pi_4 \ldots \ldots \Pi_s) \quad - \quad (2A)$$

Where all the $\Pi_i$ terms are dimensionless and independent.

Since equation (2A) is entirely general, it applies to any other system which is a function of the same variables. Hence, it applies to a specific system called the model.

$$\Pi_{1m} = F(\Pi_{2m}, \Pi_{3m}, \Pi_{4m} \ldots \ldots \Pi_{sm}) \quad - \quad (2B)$$

An equation for predicting $\Pi_1$ from $\Pi_{1m}$ may be found directly by dividing equation (2A) by equation (2B).

Therefore: $$\frac{\Pi_1}{\Pi_{1m}} = F\left(\frac{\Pi_2}{\Pi_{2m}}, \frac{\Pi_3}{\Pi_{3m}}, \frac{\Pi_4}{\Pi_{4m}} \ldots \ldots \frac{\Pi_s}{\Pi_{sm}}\right) \quad - \quad (2C)$$

Now, if the model is designed and operated so that -

$$\begin{align*}
\Pi_{2m} &= \Pi_2 \\
\Pi_{3m} &= \Pi_3 \\
\Pi_{5m} &= \Pi_5
\end{align*} \quad - \quad (2D)$$
It follows that:

\[ F (\Pi_2, \Pi_3, \Pi_4, \ldots, \Pi_s) = F (\Pi^m_2, \Pi^m_3, \ldots, \Pi^m_s) - (2E) \]

The nature of the function is identical for the model and prototype because equation (2A) is general.

From equation (2C) and (2E), it is apparent that:

\[ \Pi_1 = \Pi^m_1; \text{ this is the prediction equation.} \]

Now, rewriting equation (1):

\[ V = F (1, \lambda, A, DL, W, d, H, \rho_s, \rho_b, V_c, d_d, v, \theta, \theta) - (1) \]

To apply dimensional analysis and theory of models, Equation (1) may be written as:

\[ c_\lambda V c_1 \lambda c_2 c_3 A c_4 D c_5 W c_6 d c_7 H c_8 \rho c_9 c_{10} V c_{11} d_d c_{12} v c_{13} g c_{14} g c_{15} = 0 - (A) \]

The corresponding dimensional equation is:

\[ (L^3)c_1 L c_2 L c_3 (L^2)c_4 L c_5 (ML-2 T-2) L c_7 L c_8 (ML-3)c_9 (ML-3)c_{10} (L^3)c_{11} L c_{12} (LT-1)c_{13} (-)c_{14} (-)c_{15} = 0 \]

From the above, the auxiliary equations may be written as:

\[ (B) - M: \quad C_6 + C_9 + C_{10} = 0 \]

\[ (C) - L: \quad 3C_1 + C_2 + C_3 + 2C_4 + C_5 - 2C_6 + C_7 + C_8 - 3C_9 - 3C_{10} + 3C_{11} + C_{12} - C_{13} = 0 \]

\[ (D) - T: \quad -2C_6 - C_{13} = 0 \]

Since three equations are available for solving thirteen unknowns, arbitrary values must be assigned to ten of the unknowns, many combinations are
possible, of these one involving: $C_1$, $C_3$, $C_4$, $C_5$, $C_6$, $C_7$, $C_8$, $C_{10}$, $C_{11}$, $C_{12}$, $C_{14}$, $C_{15}$, has been selected.

The determinant of the coefficients of the remaining terms $C_2$, $C_9$ and $C_{13}$ is:

\[
\begin{vmatrix}
0 & 1 & 0 \\
1 & -3 & -1 \\
0 & 0 & -1 \\
\end{vmatrix} \neq 0
\]

Since this is not equal to zero, the resulting equations are independent and the selection is valid.

Values are assigned arbitrarily as follows:

\[
\begin{align*}
C_1 &= 1 \\
C_3 &= 0 \\
C_4 &= 0 \\
C_5 &= 0 \\
C_6 &= 0 \\
C_7 &= 0 \\
C_8 &= 0 \\
C_{10} &= 0 \\
C_{11} &= 0 \\
C_{12} &= 0
\end{align*}
\]

Substitute these values in equations (B), (C), (D)

\[
\begin{align*}
C_9 &= 0 \quad -3a \\
3 + C_2 - 3C_9 - C_{13} &= 0 \quad -3b \\
- C_{13} &= 0 \quad -3c
\end{align*}
\]

From equations 3a, 3b and 3c:

$C_2 = -3$ and from above: $C_1 = 1$
From this and equation (A) - dropping \( C_\alpha \)

\[
\Pi_1 = \frac{v_{13}}{1}
\]

From the Pi theorem, it is seen that a total of 10 Pi terms must be determined. Another term may be found by selecting a different combination of arbitrary values for the selected exponents; for example:

\[
\begin{align*}
C_1 &= 0 \\
C_3 &= 1 \\
C_4 &= 0 \\
C_5 &= 0 \\
C_6 &= 0 \\
C_7 &= 0 \\
C_8 &= 0 \\
C_{10} &= 0 \\
C_{11} &= 0 \\
C_{12} &= 0 \\
\end{align*}
\]

Substitute above in equations (B), (C), (D)

\[
\begin{align*}
C_9 &= 0 -4a \\
C_2 + 1 - 3C_9 - C_{13} &= 0 -4b \\
- C_{13} &= 0 -4c \\
\end{align*}
\]

From Equations 4a, 4b, 4c:

\[
C_2 = -1; \quad \text{and from above: } C_3 = 1
\]

From this and equation (A) dropping \( C_\alpha \) or:

\[
\Pi_2 = \frac{\lambda_i}{1} \quad 11
\]
Another Pi term may be found by letting $C_4 = 1$ with other selected exponents equalled to zero:

$$\Pi_3 = \frac{A}{12} \quad \text{III}$$

Similarly, seven more independent Pi terms are developed by letting $C_5, C_6, C_8, C_9, C_{11}, C_{12}$ and $C_{13}$ in turn, equal unity, with the other selected exponents equal to zero, thus:

$$\Pi_4 = \frac{D}{I} \quad \text{IV}$$

$$\Pi_5 = \frac{w.l}{\rho_s v^2} \quad \text{V}$$

$$\Pi_6 = \frac{d}{I} \quad \text{VI}$$

$$\Pi_7 = \frac{H}{I} \quad \text{VII}$$

$$\Pi_8 = \frac{\rho_b}{\rho_s} \quad \text{VIII}$$

$$\Pi_9 = \frac{V_c}{13} \quad \text{IX}$$

$$\Pi_{10} = \frac{dd}{I} \quad \text{X}$$

And also for the dimensionless variables $\phi, \theta$:

$$\Pi_{11} = \phi \quad \text{XI}$$

$$\Pi_{12} = \theta \quad \text{XII}$$

A general solution may, therefore, be written as:

$$\frac{V}{13} = F(\frac{\lambda_i}{1}, \frac{A}{12}, \frac{D}{I}, \frac{w.l}{\rho_s v^2}, \frac{d}{I}, \frac{H}{I}, \frac{\rho_b}{\rho_s}, \frac{V_c}{13}, \frac{dd}{I}, \phi, \theta) \quad \text{(5)}$$
APPENDIX II

TABLES I and IA - RESULTS.
TABLE 1 - (LONGITUDINAL SUB LEVEL CAVING)

<table>
<thead>
<tr>
<th>SERIAL NO.</th>
<th>TEST NO.</th>
<th>ORE BODY WIDTH</th>
<th>F.W. ANGLE</th>
<th>BLAST-RETREAT DISTANCE</th>
<th>MATERIAL IN MODEL</th>
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<th>ORE MATERIAL TYPE</th>
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<th>LOADING PATTERN</th>
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<td>10048 144 144 144 90.8 37.1 85° F.W.SLASH</td>
<td>10048 144 144 144 90.8 37.1 85° F.W.SLASH</td>
<td>10048 144 144 144 90.8 37.1 85° F.W.SLASH</td>
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<td>10048 144 144 144 90.8 37.1 85° F.W.SLASH</td>
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V* - VOLUME OF HEAPED BUCKET OF SCOOP - 112  CU.FT.
N** - DRAW FIGURES PLOTTED AND COLUMNS 16, 17, 18 CALCULATED FOR NO. OF SCOOPS DRAWN 'N'.
SC - SCOOPS DRAWN.
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<p>| 15| 28 | 20°| 65°| 45°| 7° | 1.68| 1.60| 1.35| A  | &quot;                           | 6405| 92 | 92 | 140 | 86.3| 38.2| &quot;                           |
| 16| 28 | 20°| 65°| 45°| 6° | 6° | 1.68| 1.60| 1.35| A  | &quot;                           | 5490| 78 | 78 | 114 | 80.5| 39.4| &quot;                           |
| 17| 28 | 20°| 65°| 30°| 8° | 1.68| 1.60| 1.35| A  | &quot;                           | 4536| 65 | 65 | 124 | 66.3| 46.5| &quot;                           |
| 18| 28 | 20°| 65°| 30°| 6° | 6° | 1.68| 1.60| 1.35| A  | &quot;                           | 3402| 49 | 54 | 131 | 73.0| 44.6| &quot;                           |
| 19| 37 | 20°| 65°| 60°| 10°| 1.56| 1.56| &quot;   | F  | &quot;                           | 12560| 173| 191| 120 | 81.9| 28.3| 80°F.W.SLASH               |
| 20| 37 | 20°| 65°| 60°| 8° | 1.56| 1.56| &quot;   | F  | &quot;                           | 10048| 140| 154| 121 | 86.1| 28.4| &quot;                           |
| 21| 37 | 20°| 65°| 60°| 8° | 1.56| 1.56| &quot;   | F  | &quot;                           | 10048| 140| 167| 134 | 87.1| 34.2| &quot;                           |
| 22| 37 | 20°| 65°| 60°| 7° | 1.56| 1.56| &quot;   | F  | &quot;                           | 8792 | 121| 135| 121 | 82.3| 31.8| &quot;                           |
| 23| 37 | 20°| 65°| 60°| 6° | 1.56| 1.56| &quot;   | F  | &quot;                           | 7536 | 103| 115| 125 | 79.7| 35.9| &quot;                           |
| 24| 37 | 20°| 65°| 60°| 5° | 1.56| 1.56| &quot;   | F  | &quot;                           | 6280 | 86 | 96 | 124 | 74.0| 40.5| &quot;                           |
| 25| 37 | 20°| 65°| 45°| 8° | 1.56| 1.56| &quot;   | F  | &quot;                           | 7520 | 104| 115| 124 | 83.5| 32.5| &quot;                           |
| 26| 37 | 20°| 65°| 45°| 6° | 1.56| 1.56| &quot;   | F  | &quot;                           | 5395 | 74 | 82 | 115 | 81.1| 29.2| &quot;                           |
| 27| 37 | 20°| 65°| 30°| 8° | 1.56| 1.56| &quot;   | F  | &quot;                           | 5024 | 70 | 77 | 122 | 63.6| 43.2| &quot;                           |
| 28| 37 | 20°| 65°| 30°| 6° | 1.56| 1.56| &quot;   | F  | &quot;                           | 3360 | 46 | 51 | 118 | 66.7| 43.2| &quot;                           |
| 29| 37 | 20°| 65°| 30°| 5° | 1.56| 1.56| &quot;   | F  | &quot;                           | 2800 | 39 | 43 | 120 | 59.3| 50.5| &quot;                           |
| 30| 35 | 20°| 65°| 60°| 8° | 1.56| 1.56| &quot;   | E  | &quot;                           | 10048| 140| 154| 112 | 74.9| 33.0| 80°F.W.SLASH               |
| 31| 35 | 20°| 65°| 60°| 6° | 1.56| 1.56| &quot;   | E  | &quot;                           | 7536 | 104| 115| 113 | 71.2| 37.1| 1:20 SCALE TEST            |
| 32| 35 | 20°| 65°| 45°| 6° | 1.56| 1.56| &quot;   | E  | &quot;                           | 5652 | 72 | 79 | 113 | 71.3| 37.1| &quot;                           |
| 33| 35 | 20°| 65°| 30°| 8° | 1.56| 1.56| &quot;   | E  | &quot;                           | 5024 | 70 | 77 | 107 | 54.0| 49.1| &quot;                           |
| 34| 35 | 20°| 60°| 60°| 8° | 1.56| 1.56| &quot;   | B  | 0-230 SC = 1:3              | 15520| 199| 219| 13  | 82.5| 18.0| NO F.W.SLASH                |</p>
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| 93 | 33 | 50' | 55° | 60' | 8'-4' | 1.46 | 1.50 |   |   |   |   | 23112 |   | 110 | 60.1 |
| 94 | 33 | 50' | 55° | 60' | 8'-6' | 1.46 | 1.50 |   |   |   |   | 25356 |   | 112 | 65.8 |
| 95 | 33 | 50' | 55° | 60' | 6'-6' | 1.46 | 1.50 |   |   |   |   | 20700 |   | 112 | 68.4 |
| 96 | 33 | 50' | 55° | 60' | 6'-4' | 1.46 | 1.50 |   |   |   |   | 18456 |   | 111 | 61.7 |</p>
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<td>6' 1.80</td>
<td>1.80</td>
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</table>

**V** - VOLUME OF HEAPED BUCKET OF SCOOP - 112 CU.FT.

**N** - DRAW FIGURES PLOTTED AND COLUMNS 16, 17, 18 CALCULATED FOR NO. OF SCOOPS DRAWN 'N'.

SC - SCOOPS DRAWN.
APPENDIX III

TABLES 2, 3 AND 4
<table>
<thead>
<tr>
<th>TEST NO.</th>
<th>DESCRIPTION OF TESTED MATERIAL</th>
<th>% MOISTURE</th>
<th>DIAL READING</th>
<th>PEAK AXIAL STRESS (σ)</th>
<th>CONFINING PRESSURE</th>
<th>ANGLE OF INTERNAL FRICTION (φ₁)</th>
<th>ANGLE OF SLIDING (45°-φ₂)</th>
<th>REMARKS</th>
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<td>1</td>
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<td>-</td>
<td>130</td>
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<td>1.22</td>
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<td>-0.525&quot;</td>
<td>-0.371&quot;</td>
<td>-0.263&quot;</td>
<td>-0.185&quot;</td>
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<td>-0.0328&quot;</td>
<td>-0.0232&quot;</td>
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<td>+0.185&quot;</td>
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<td>14.00%</td>
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<td>35.03%</td>
<td>11.20%</td>
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<td>3.00%</td>
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<tr>
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<td>6.62%</td>
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<td>15.90%</td>
<td>15.12%</td>
<td>27.55%</td>
<td>7.97%</td>
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<td>32.73%</td>
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<td>13.69%</td>
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<td>25.0</td>
<td>10.0</td>
<td>1.46</td>
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### APPENDIX IV

**TABLE 5 - DESIGN DATA FOR THE RECOMMENDED LAYOUTS.**
| SERIAL NUMBER | CONFIGURATION | ORE BODY WIDTH | F.W. ANGLE | SUB LEVEL INTERVAL | BLAST RETREAT DISTANCE | TOTAL EXTRACTION (%) | ORE RECOVERY (%) | WASTE DILUTION (%) | LOADING PATTERN | EXTRATION DRIFT | H.W. :: F.W. | F.W. SLASH | ITEM NO. - FROM TABLE I and 1A |
|---------------|---------------|----------------|------------|--------------------|------------------------|----------------------|-----------------|----------------|----------------|----------------|---------------|------------|----------------|----------------|
| 1             | A             | 20'            | 65°        | 60'                | 8'                     | 126                  | 86.1            | 28.4           | 0-90 SC = 0::1 | 80°            | 20           |            |                |                |
|               |               |                |            |                    |                        |                      |                 |                |                |                |              |            |                |                |
| 2             | A             | 20'            | 65°        | 60'                | 8'                     | 112                  | 74.9            | 33.0           |                | 80°            | 30 (1:20 SC) |            |                |                |
| 3             | B             | 30'            | 65°        | 60'                | 8'                     | 124                  | 89.3            | 28.0           | 0-90 SC = 0:1 | 80°            | 49           |            |                |                |
|               |               |                |            |                    |                        |                      |                 |                |                |                |              |            |                |                |
| 4             | F             | 30'            | 75°        | 60'                | 8'                     | 121                  | 87.4            | 27.9           | 0-110 SC = 0:1 | N/A            | 60           |            |                |                |
|               |               |                |            |                    |                        |                      |                 |                |                |                |              |            |                |                |
| 5             | C             | 40'            | 65°        | 60'                | 8'                     | 115                  | 90.5            | 21.3           | 0-180 SC = 0:1 | 80°            | 78           |            |                |                |
|               |               |                |            |                    |                        |                      |                 |                |                |                |              |            |                |                |
| 6             | D             | 50'            | 55°        | 60'                | 6'- 6'                 | 112                  | 68.5            |                |                | 180-300 SC = 1:1 |            | 95          |                |                |
| 7             | E             | 50'            | 70°        | 60'                | 8'- 6'                 | 110                  | 78.4            |                |                | 180-300 SC = 1:1 |            | 98          |                |                |
| 8             | H             | T.S.L.C.       | 79°        | 30'                | 6'                     | 109                  | 87.3            | 20.1           | 0-200 SC = 1:1 | 80°            | -            | 104         |                |                |
CONFIGURATION - A

STANDARD LAYOUT FOR 20-FT. ORE BODY - F.W. ANGLE 65° (TO 75°) (ITEM NO. 20 AND 30 TABLE 1 OF APPENDIX III)

A - MIN. 12'; RECOMMENDED WIDTH
B - MAX. 16'; TO SUIT SPECS. OF PRODUCTION JUMBO IN ORDER TO EQUALIZE THE WORK LOAD ON TWO BOOMS.
CONFIGURATION B

STANDARD LAYOUT FOR 30 FT. ORE BODY - F.W. ANGLE 65° (TO 75°)
ITEM NO. 49 TABLE I OF APPENDIX II)
CONFIGURATION C

STANDARD LAYOUT FOR 40 FT. ORE BODY - F. W. ANGLE 65° (TO 75°)
(ITEM NO 78 TABLE 1 OF APPENDIX 11)
CONFIGURATION D

STANDARD LAYOUT FOR 50 FT. ORE BODY - F.W. ANGLE 55°
(ITEM NO. 95 TABLE 1 OF APPENDIX II)

VERTICAL SURFACE OF SLIDING

DRIFT 'B'

12'

27'

DRIFT 'A'

12'
CONFIGURATION E

STANDARD LAYOUT FOR 50 FT. ORE BODY - F. W. ANGLE 70°
(ITEM NO. 98 TABLE 1 OF APPENDIX II)
CONFIGURATION F

STANDARD LAYOUT FOR 20 FT., 30 FT. AND 40 FT. ORE BODY - F. W. ANGLE 75° AND ABOVE.
CONFIGURATION G

STANDARD LAYOUT FOR 20 FT., 30 FT. AND 40 FT. ORE BODY - F. W. ANGLE 55° - 65°.
CONFIGURATION H:

STANDARD LAYOUT FOR TRANSVERSE SUB LEVEL CAVING
SUB LEVEL INTERVAL 30 FT. - SIDE SLOPES 79°
(ITEM NO 103 TABLE 1B OF APPENDIX II)
CONFIGURATION I

STANDARD LAYOUT FOR TRANSVERSE SUB LEVEL CAVING
SUB LEVEL INTERVAL 30 FT - SIDE SLOPES 79°

VERTICAL SECTION ALONG THE CENTRE LINE OF THE EXTRACTION DRIFT FROM CONFIGURATION 'H'
CONFIGURATION J

COMPARISON OF LOADING INTENSITY

30 FT. ORE BODY - F. W. ANGLE 65°
LOADING INTENSITY 3:1 :: F.W. : H.W.
(ITEM NO. 41 TABLE 1 OF APPENDIX II)
CONFIGURATION K

COMPARISON OF LOADING INTENSITY

30 FT. ORE BODY - F. W. ANGLE 65°
LOADING INTENSITY 1:1 :: F.W. : H.W.
(ITEM NO. 40 TABLE 1 OF APPENDIX II)
TABLE 6.

Test No. 33 (#85 to #88 of TABLE 1)

Cumulative number of scoops drawn are tabulated below:

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<td>H.W.</td>
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<tr>
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<td>-</td>
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<td>251 - 350</td>
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TABLE 7.

Test No. 32 (#89 to #92 of TABLE 1)

Cumulative number of scoops drawn are tabulated below:

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<th>DRIFT B</th>
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<tbody>
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<td>-</td>
<td>-</td>
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<tr>
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<td>366 - 370</td>
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<td>396 - 400</td>
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</table>
FIGURE 14  Longitudinal Sub Level Caving - shows sequence of draw with single extraction drift on each successive sub level. Ore body width 20', S.L.I. = 60', F.W. angle 65° and F.W. slash 80°.

(PP 120 to 124)

FIGURE 15  Longitudinal Sub Level Caving - shows sequence of draw with two extraction drifts on a sub level. Orebody width 50', S.L.I. = 60', F.W. angle 55° and F.W. slash 80°.

(PP 125 to 127)
Appendix VI

Detailed Geology

Granduc Mines' ore occurs in long, irregular conformable tabular bodies with an average width of 40', combined strike length of 3,500' and vertical extent of about 2,500'. These lenses are composed of stringers and disseminations of pyrrhotite, chalcopyrite and magnetite. They lie in a series of weakly metamorphosed and strongly folded phyllonites (silicious metasediments), in a group called the mine member, which is about 500' thick between the andisitic footwall and thin lime stone hanging wall, which is succeeded by altered greywackes. The structural ore control is folds, faults and sheared crumpled zones.

The orebodies are generally dipping at 70° although some parts dip at 50 - 55° and others at 90°. Thick portions in "C" orebody are up to 120 feet wide, whereas the wings of "C" orebody and other A, B and B1 orebodies vary between 15 feet to 50 feet in width. Almost 50% of the total minable tonnages in No. 1 Mining Zone are tied up in thin portions.

Geological ore reserves (1966) are 43 x 10^6 tons at average 1.73% cu.
With 10% dilution, ore reserves are approximately 47 x 10^6 tons at 1.49% cu.

A typical section and plan of the orebodies are attached on Page 132.