Design and Operational Issues for
Increasing Sublevel Cave Intervals at Stobie Mine

by

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in conformity with the requirements for
the degree of Master of Applied Science

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ABSTRACT

The objective of this study was to provide practical insight into the design and operation of sublevel cave stopes. In particular, the effects of increased sublevel intervals were investigated in order to understand their impact upon stope recovery. The investigation included; review of the flow theory for caving rock, physical model tests and underground mine observation.

The technical and financial support for this study was provided by INCO Ltd., where the plan is to increase sublevel intervals from 21 to 31 metres. The implications of design to the recovery of the sublevel cave stopes, with the greater sublevel intervals, is described in this thesis. Practical design strategy is discussed with the Sliding Plane (SP) concept and recommendations for the mine operations are included.

The physical model tests indicate that strong vertical flow is necessary for high recoveries for the 31 m sublevel intervals. The vertical flow component is primarily influenced by steep ring gradients and wide drawpoints. The geometrical and operational parameters involved with the sublevel cave mining process should promote maximizing both these parameters.

Results from the underground operations concluded that fragmentation is of key importance to sublevel cave stope. Proper fragmentation of the ore column resulted in improved efficiency for the other mine operations and was critical to recovery. Underground observations indicated that the drawpoints must be wide enough to account for inconsistent blast results and provide regular flow. The drawpoint width determines the width and flow strength of the central channel, which is important to achieving high recovery of the ring.

The high mobility and lack of particle interlock are the main differences between the flow regime in the model and the underground sublevel cave stope. The model results indicated that 31 metre sublevel intervals are viable for the sublevel cave design provided that burdens are increased accordingly. The larger burdens may not be possible for the underground sublevel cave design due to poor fragmentation and subsequent low particle mobility within the stope.
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1.0 INTRODUCTION

The fundamental purpose of this research is to provide insight into the gravity flow characteristics of blasted rock in a sublevel cave mining operation, as these characteristics affect the viability of increasing sublevel intervals. This investigation involved analyzing data from physical test models and actual mine results in order to develop a plan to optimize sublevel cave mining layouts. The technical and financial support for this study was provided by International Nickel Company Limited (INCO). The design strategy at INCO's Stobie Mine was to increase the sublevel interval from approximately twenty-one to thirty-one metres. These increased level distances are intended to reduce overall development costs when mining sublevel cave stope areas.

Understanding the principles of gravity flow is central to optimizing design and operating practices in sublevel cave mining. Despite its significance, Brady and Brown (1993) conclude that the mechanics of gravity flow of blasted or caved ore is not well understood by mining engineers. To date, the most widely accepted theories of the gravity flow of solids are from Kvapil (1965). Gravity flow of blasted ore and caved waste in sublevel caving is a process more complicated than that which is used in bin flow. The geometry, dimensions, and even operational constraints of sublevel caving cannot be selected at random but must be planned while respecting the characteristics of gravity flow of coarse materials. The laws of gravity flow are independent of rock size. However, the particle size distribution, intrinsic properties of the rock and the internal stresses within the stope do play a part in dictating flow behaviour.

1.1 Study Objectives

The design for sublevel cave operations has traditionally been related to the bulk flow of solids in bins or hoppers. Mining engineers would apply the flow behaviour of bulk solids to their mine design while allowing for appropriate draw conditions and production schedules. Unfortunately, the realities of the underground environment provide further challenges with respect to stress and operational inconsistencies which make theoretical solutions impractical for
optimizing stope design. The thesis presented herein will attempt to incorporate practical design guidelines and ideas for planning engineers while relating these concepts to bin flow theory.

The sophistication of bin flow theory, accompanied by the various flow form models, have complicated the analysis for the draw of broken rock in mining. This has perhaps reduced the popularity of the method in the view of design engineers. Sublevel cave mining is regarded as a “natural” mining method and as such is highly dependent upon the flow characteristics of the blasted rock material within the caving stope.

Defining flow characteristics solely in terms of mathematical equations is impractical due to the complex nature of the relationship between boundary conditions and drawing patterns. The drawing or flow pattern characterizes the different flow zones within the caving area. This study has investigated flow patterns which result from a single source of loosening, i.e. the drawpoint. Simplification of this type of flow pattern will assist the design engineer in the proper design of sublevel cave stopes. This thesis will attempt to clarify the connection between design changes and the resultant flow pattern.

1.2 Methodology

The study of flow behaviour in sublevel cave operations has been investigated by two methods:

1) Physical Model Tests
2) Mine Environment Observation

Details of the methodology are provided in Sections 3.2 and 4.2 respectively. The results from the investigations explore and provide insight into the flow behaviour of solids in a sublevel cave mine stope. The formation and progressive development of the various flow zones will be clarified from the observations of the models. The aim of the tests will be to make the connection between design parameters and flow behaviour; this being gleaned from the overall observation of the models
1.3 The Sublevel Cave Mining Method

Sublevel caving is a bulk mining method which is largely dependent upon the gravity flow characteristics of ore and waste rock. The lower recoveries associated with the method typically limit its use to base metal deposits of low grade. Marginal ore can be handled because of the low operating costs and because relatively high dilution and low recovery can be tolerated. Sublevel caving is not a widely practiced mining method since its successful application is largely dependent upon the grade as well as the geological and rock characteristics of the orebody.

Sublevel cave mining is used in orebodies with relatively weak but competent ore zones. In weak ore, other mining methods may not be safe. Craigmont Mine, for example, is a highly fractured and faulted, disseminated chalcopryite body with large gauge zones and was successfully mined by sublevel caving (Sandstrom, 1972). Mining situations with unstable ground conditions utilize the method since hanging wall and pillar failure found with other mining methods may be avoided. In fact, Johnston (1982) indicated that sublevel caving was initially implemented at Stobie Mine for this very reason. The progression of the open blasthole stopes to the intermediate levels of the mine caused hanging wall failure and pillar instability. The stress applied to the narrow pillars threatened the continuation of safe production. It was determined that the benefits of the low ore grade recovery could not absorb the high cost of filling in the open stopes, so sublevel caving was adopted for over half of the remaining pillars. This enabled the company to resume safe recovery with high production rates from a highly mechanized mining method.

Modern sublevel cave mining has been adapted to stronger ground and is no longer a natural caving method since ore flow is induced by explosives. The caving action of the walls, however, has preserved its name. The lower recoveries, high dilution, critical significance of drilling and blasting operations and comparatively high development costs for sublevel cave have decreased its popularity as a mining method. Currently, planning engineers seem to prefer mining methods that use artificial support which simplifies mine design and production scheduling rather than a method such as sublevel cave which does not allow as direct a control over these operational requirements.
The most distinct feature of sublevel cave is that the method does not require the use of backfill or pillars for its implementation. The caved rock material fills in the void of the open stope thus forming a natural support system for the mining area. This feature is particularly important for situations which require large-scale pillar recovery or where backfill is prohibitively expensive or operationally infeasible.

A typical sublevel caving operation (Fig. 1), consists of a series of sublevels driven from the footwall to the hangingwall. These sublevels serve to cut the orebody into a series of slices which are then mined from top to bottom. The sublevels are developed from a number of ore passes which have been raised from the main haulage drift. The main haulage drift is found in the footwall or hangingwall, depending upon the direction of mining retreat within the stoping area. The main haulage drift is usually parallel to the strike of the dominant vein structure. From the haulage drifts, a series of production drifts or crosscuts are driven into the orebody from one side to the other of the orebody. These production tunnels are called drifts if they run parallel to the orebody and are called crosscuts if they intersect the body perpendicularly. It is generally accepted that crosscuts are used for wide orebodies in transverse sublevel caves while drifts are used for narrow veined structures in longitudinal sublevel caves.

Reference to Section B-B' of Figure 1 shows that the production tunnels or crosscuts on successive sublevels are staggered. This implies that ore drawn from a sublevel can originate from two sublevels above. The production tunnels are also evenly spaced throughout the orebody thus dividing the deposit into a geometrical pattern. The dimensions of the production tunnels are mainly dependent upon the rock mass strength and existing stress regime as well as the size of equipment being used during operation. The current trend is to increase the size of these tunnels by 50 to 70 percent over the older designs and to increase the stoping area for that drift by up to 500%. This effectively minimizes development costs and time while keeping the dimensions of these production tunnels to a minimum with respect to the stope size. The new sublevel cave designs reduce development ore to less than 7 percent of stope production thus increasing the economic advantage of the stope.
Prior to actual production blasting, a slot is usually developed at the far end of the orebody by drilling and blasting to the sublevel overhead. The slot is blasted and mucked in individual slices to allow for the swell of the blasted ore. These slices are blasted upwards until the waste cap or pit muck from the overhead level begins to enter the extraction drift. The ore is then ring drilled in a predetermined pattern from the production tunnels in an upward direction. Blasting of the rings proceeds from the caved slot toward the haulage drift.

After a ring or series of rings has been blasted, the broken ore caves into the production tunnel, where it is loaded and transported to an orepass. As the ore caves and flows into the production tunnel the waste wall rock continuously caves on top of the blasted ore. This results in dilution to the caving ore; the dilution increasing as more muck is extracted from the heading.
Dilution of the ore is inevitable with this method and complete recovery is therefore infeasible since the ore and waste rock mix during caving.

Sublevel cave mining requires high powder factors of approximately 0.5 to 1 kg/tonne (Preston, 1996). Ore from the ring is not thrown to an open void area but is actually shattered in place. The 'void' volume for sublevel cave consists of the 10 to 20% open space contained within the unconsolidated rockfill. Therefore mines using this method may always be forced to overblast in order to prevent benching in the ore column. Benches are un-fragmented sections of the ring. The expansionary force of the gas produced by blasting is also important if room is to be provided for the swell of the blasted ore against the unconsolidated rockfill. These factors lead to the conclusion that large burdens may not be suitable for sublevel cave stope blasts.

The successive rings in the production tunnel are usually drilled off before loading operations commence. Loading is typically done with pneumatic-type portable loaders with ANFO being preferred in dry holes while emulsion or bulk slurry type explosives are required in wet holes. The competence of the brow is maintained by inclining blast rings to the opposite direction of mining retreat and by not fully loading the holes so that powder factors at the collar may be reduced. Control of the brow prevents drawpoints from collapsing and allows for safe access to be maintained when loading operations resume. Ore loading and transporting is done with Load-Haul-Dump (LHD or Scooptram) units and a combination of LHDs and trucks if haulage distances are excessive. Uniform mucking across the entire width of the drawpoint induces a more even draw down movement of the blasted ore which in turn improves recovery. LHD operators may have to adopt a side to side mucking procedure if the width of the bucket is less than the width of the drawpoint.
1.4 Advantages & Disadvantages of Sublevel Cave Mining

The advantages and disadvantages of sublevel cave mining can be summarized as follows:

**Advantages**

1. **Low Cost:** Fill is not required and manual labour requirements are low.
2. **Increased Safety:** Miners are not exposed to open stope areas. The mine cycle is consistent which lends itself to safe mine practice.
3. **Flexible Mine Operations:** The method is suitable for a wide variety of orebodies which may vary in shape, size, ore competency and grade distribution. Mine production occurs concurrently with development. Development ore represents 7% to 20% of stope production.
4. **Increased Efficiency:** The mining process is uniformly applied and adaptable to mechanization. Improvements in mining equipment and automation may be easily integrated to this method thereby improving its overall efficiency. Pillars are not left after the mining cycle has been completed. Pillars constitute a loss of ore.
5. **Increased Production:** High production rates possible due to the large number of drawpoints.

**Disadvantages**

1. **Low Recoveries:** compared to other methods. Recovery can range from 75% to 90%. Low recovery makes the method uneconomical for high-grade deposits with high in situ value. This generally rules out precious metal deposits where recovery must be close to 100%.
2. **High Dilution:** up to 40% can be encountered.
3. **Complex Ventilation:** Extraction headings require vent tubing since fresh air cannot be pulled through the caved stope rock. The dead-end production tunnels may complicate ventilation requirements and planning.
4. **Mine Instability:** Progression of mining to deeper levels may create instability in the pillars between production tunnels. If the pillar widths are increased to prevent this failure, then lower recovery can occur. Greater depths, past 1100 metres, are generally inadvisable for sublevel cave since the flow of the broken ore will become restricted due to the increasing confining field stresses around the blasted ring pattern. Surface subsidence may also pose a problem since it is likely to occur. Consideration must be given to surface conditions and ground water locations.

5. **Mine Scheduling and Development:** The relatively complex mine design and development scheduling can be a major deterrent to implementing this mining system. Development work for the method is slow, costly and substantial. The development costs must also be paid for early in the life of the mine.
2.0 PRINCIPLES OF ORE FLOW IN SUBLEVEL CAVING

The problem of analyzing the progressive flow of rock through an enclosed stope is not wholly analogous to the theories developed for the flow of other granular solids. Yenge (1980) contends that the flow models developed for other fine grained granular solids, such as sand and powders, are not directly applicable. This is due to the discrepancies in the particle sizes, the relative sizes between the containers and particles and the boundary conditions within the respective environments. There are also appreciable differences between the slower discharge rates, the large increases in void volume and the delayed ground caving response in the mining problem as compared to granular solid models. Despite these problems, ore flow has historically been related to the motion of granular material in a bunker. Three distinct differences exist between the flow of material in bunkers and in the sublevel caving system. These can be summarized as follows:

1. The friction between the broken rock and the solid face of the unbroken ring affects the flow pattern of the blasted material. Material in a bunker is surrounded by four solid walls whereas the rock in the sublevel caving system is surrounded on three sides by broken rock and on the fourth side by the un-blasted ring.

2. Blasting the ore column tends to create density variations between the ore and waste material in the sublevel caving system.

3. Sublevel caving exists under substantially higher over-burden pressures than are usually found in bunker flow.

The theory proposed by Janelid and Kvapil (1966) continues to be the most comprehensive and suitable definition for the theory of gravity flow and its applicability to the sublevel caving mining system. Their results are based on a case of free discharge of granulated material through an outlet at the bottom of a bunker. Central to this theory is the expansion of a flow ellipsoid which is developed as rock within the stope is extracted.


2.1 Development of the Flow Ellipsoid

As the broken ring is extracted, the progression of the draw channel forms into distinct zones of movement which vary in mobility and vector velocity. The combination of these zones forms an elliptical flow pattern which does not have the exact shape of an ellipsoid but for ease of reference and analysis will be referred to as such. The particles within the ellipse are subject to two rudimentary forms of motion: direct and indirect.

Direct Motion: This type of motion occurs immediately above the effective opening of the drawpoint. It may also be referred to as the zone of mass flow since the particles remain in their respective positions within the flow assemblage during movement to the drawpoint.

Indirect Motion: This motion incorporates the sliding and rotational displacements which occur as a result of the direct movements. This secondary motion occurs as the particles in the mass flow zone move out of place and are replaced by adjacent particles.

When the ring is blasted and flow is allowed to occur, all the immediately discharged material originates from an ellipsoidal zone known as the motion ellipsoid. This zone is most strongly subjected to direct motion because it is directly above the drawpoint opening. The material between the motion ellipsoid and the limit ellipsoid is subject to direct and indirect forces where the indirect motion occurs to the particles between the motion ellipsoid and the limit ellipsoid. Beyond the limit ellipsoid the particles remain stationary in a region known as the passive zone while the area within the limit ellipsoid is known as the active zone. The location and size of the respective zones are illustrated in Figure 2. As drawing proceeds, the material within the motion ellipsoid is removed and replaced by surrounding particles. It is only the material within the limit ellipsoid, however, that has the opportunity to enter the motion ellipsoid area. The size and the eccentricity of the motion and limit ellipsoids gradually develop as material is removed from the stope. Upon completion of the stope, the limit ellipsoid is typically 14 to 16 times larger than the motion ellipsoid.
The aforementioned ellipsoids expand as rock is removed from the drawpoint. The limit ellipsoid develops as recharge material from the surrounding rock enters the motion ellipsoid as material is drawn from the discharge point. Taylor (1972) described the following:

The ellipsoid of motion is essentially an envelope bounding the source of all material removed up to some particular moment in time. A much larger surrounding ellipsoid (limit ellipsoid) defines the limit of broken material that, up to that same moment, has expanded and moved to refill the draw ellipsoid. It follows that any instantaneous draw ellipsoid is merely one of an expanding series that, if continued far enough, could lie within an open-topped paraboloid of expansion reaching the ground surface.
Physical sand flow models from Kvapil (1965) indicate that the draw down of material is in the form of an inverted cone. This indicates that vector velocity in the center of the draw is highest and is reduced proportionally on either side of the draw cone axis until a particle velocity of zero is achieved at the boundary of the extraction ellipsoid. The velocity distribution is shown in Figure 3, which represents the velocity distributions through sections E-E' to A-A'. The boundaries of the limit ellipsoid represent an instantaneous velocity of zero and the central flow axis vectors indicate the progression of relative values such that $V_4 > V_3 > V_2 > V_1$. Note that the vector velocities are drawn perpendicular to the ellipsoid’s actual flow movement for clearer visualisation.

![Figure 3 - Velocity Distribution within the Limit Ellipsoid (Kvapil, 1992)]
2.2 Shape and Size of the Flow Ellipsoids

Flow ellipsoids are usually described in terms of their eccentricity and size. Figure 4 shows a vertical slice through a stope block as well as the major (\(a_n\)) and minor (\(b_n\)) semi-axes of the motion ellipsoid.

Figure 4 - Flow Ellipsoid Concept
Kvapil (1992) has established a simplified theory of gravity flow which defines explicit relationships between the discharged material and the motion and limit ellipsoids. This relationship maintains that, assuming approximately equal eccentricity values for both the motion and limit ellipsoids, the volume of the limit ellipsoid is fifteen times greater than that of the motion ellipsoid such that:

\[ V_{\text{le}} \approx 15 \times V_{\text{me}} \quad \text{(Eq. 1)} \]

therefore;

\[ H_{\text{l}} \approx 2.5 \times H_{\text{m}} \quad \text{(Eq. 2)} \]

By this definition, the height of the limit ellipsoid \((H_l)\) is two and a half times the height of the motion ellipsoid \((H_m)\). The height of the motion ellipsoid is significant since it represents the maximum recoverable height of pure ore. Further extraction beyond this point involves diluting the muck with waste which typically originates from the centre section of the ellipsoid.

The shape of the motion ellipsoid is described by its eccentricity \(\varepsilon\) as follows:

\[ \varepsilon = \frac{1}{a_n} \sqrt{d_n^2 - b_n^2} \quad \text{(Eq. 3)} \]

Increasing the value of the semi-major axis \((a_n)\) while holding the value of the semi-minor axis \((b_n)\) increases the value of eccentricity \(\varepsilon\) so that it approaches the value of one.

The eccentricity of an ellipsoid of motion for any material is not constant. As the average particle size varies for that material, so will its eccentricity. Larger sized particles will produce a wider ellipsoid of motion and thus a lower eccentricity. Conversely, smaller particles will generate thinner ellipsoids of motion with proportionally higher eccentricity. Therefore, it is conceivable that without uniform particle distribution within the ring, the ellipse will not have a perfectly formed shape but will widen out in areas of larger particle size. This would explain the generally accepted ‘hot-air’ balloon shape as shown in Figure 5. In theory, the upper and lower halves of the ellipsoid should be identical. In reality, the wider sections of the ellipsoid are near the toe area where the likelihood of proper fragmentation and powder factors are decreased. Practical values of \(\varepsilon\) are usually within the range of 0.92 to 0.96 for typical sublevel cave operations.
The overall eccentricity of both the limit and motion ellipsoids will increase as the height of draw increases. This factor is significant in this study since the main objective of this research has been to increase the sublevel interval by over 40 percent. Other factors which will affect the eccentricity in sublevel cave include:

- Particle shape & form (spherical, irregular)
- Particle surface roughness
- Particle friction angle and lubrication within the draw column
- Particle material properties (density, strength, moisture content)
- Rate of draw (fast, slow, continuous, interrupted)

Figure 5 – Modified Draw Shape
The combination of these factors results in developing a mobility factor for the particles within the blasted ore column. Particle mobility is not easily quantified in absolute terms but is more commonly described in qualitative terms (high-low) to allow for comparisons in flow behaviour. High mobility results in producing highly eccentric flow ellipsoids and stronger gravity flow. Low mobility indicates broader ellipsoids which are more dependent upon indirect motion than their slimmer counterparts.

The eccentricity has a direct relation to the volume contained within the motion ellipsoid. If we consider \( V_{me} \) as the volume of material contained within the motion ellipsoid of height \( H_m \), then the equation for the semi-minor axis \( b_n \) of that ellipsoid can be defined as:

\[
b_n = \frac{H_m}{2} \sqrt{1 - \varepsilon^2} \quad \text{(Eq. 4)}
\]

The value of the semi-minor axis \( (b_n) \), may indicate the optimal ring burden for a defined ellipsoid.

Eccentricity of the ellipsoid is largely dependent upon draw height and particle size and density. According to elliptical flow theory, the greatest width of the ellipse occurs at approximately the mid-height of the limit ellipsoid. At this point, it is useful to determine the width of the limit ellipsoid so that the value of the ring burden can be found. The burden is defined as half of the ring’s width at height \( H_m \). This value \( r \) is a measure of the radius of the limit ellipsoid at height \( H_m \) and can be defined as:

\[
r = \left[ H_m (H_m - H_i) (1 - \varepsilon^2) \right]^{1/2} \quad \text{(Eq. 5)}
\]

It is important to note that the equations developed in this section are relatively simplistic in that they assume a symmetrical ore flow about the central axis of the ellipsoid. This condition is probably rarely true in actual mining situations since boundary conditions in these situations are in a state of flux where the boundaries are being moved outwards as extraction continues. The discontinuous nature of rock extraction in the mine setting further complicates defining ellipsoid bounds.
Assuming that equations 1 to 5 are valid, and considering that \( H_m = 2 \times S_i \); a new expression for \( r \) can be defined as:

\[
r = S_i \, [6(1-\varepsilon^2)]^{1/2} \tag{Eq. 6}
\]

where: \( S_i \) indicates the sublevel interval and \( \varepsilon \) is the eccentricity of both limit and motion ellipsoids.

The value of \( r \) may also be thought of as the semi-width of flow for the ellipsoid at height \( H_m \). With this value, the optimum burden width (OB) can be related to the value of \( r \) such that:

\[
OB \geq r
\]

or

\[
OB \geq S_i \times (1-\varepsilon^2) \tag{Eq. 7}
\]

Maintaining this inequality will reduce ore loss and prevent premature waste dilution of the ore column. If the value of OB is much greater than the inequality, then lower recoveries should be expected. Equation 7 describes a suitable inequality for the burden thickness in relation to the sublevel interval.

It is important to note that these equations are applying theoretical knowledge, based largely on small scale laboratory tests which are thought to provide adequate insight into evaluating material flow characteristics of ore. It is advisable that these expressions be used as a guide rather than a fixed rule in determining appropriate mine designs in the underground environment.
2.3 **Design Principles based on Ellipsoid Dimensions**

The geometrical properties of sublevel cave modelling for this report will be defined in accordance with Figure 6 as follows:

*Figure 6 – Sublevel Cave Geometrical Properties*
where; $A =$ stope slice width  \hspace{1cm} P =$ pillar width between drawpoint headings  
$S_i =$ sublevel interval  \hspace{1cm} H =$ extraction drift height  
$B =$ extraction drift width  \hspace{1cm} \begin{equation} \nonumber H_i = \text{height of the limit ellipsoid (2.5 X Hm)} \end{equation}$

The interrelationship between the mentioned geometrical parameters and the rock properties and size distribution of the blasted fragments is so complex that theoretical equations do not necessarily provide the most effective designs. This point becomes clear if we accept the notion that blasted rock is a very heterogeneous material. It is believed that the effects of blasting, even with optimum drilling and blasting operations, would create zones of variously sized particles within the blasted column. These difficulties would result from the explosive alone and at this point, the effects of adverse geological conditions, such as faults, slips, and intrusions, are not even entered into the picture. The pockets of varied size ore ultimately disrupt the gravity flow of the ore particles. This typically occurs near the brow since this is the area of constricted flow and the region where compaction and settlement of small particles plays its most significant role. These factors alone indicate that ellipsoid flow in sublevel cave operations is likely to be irregular, making it impossible to find an optimal design based on simple geometrical properties within the layout. Nevertheless, the equations have proven useful in developing baseline data for the proper sublevel layout.

Sublevel layout planning may only be optimized by individualized tests at the mine in question. In order to minimize dilution, the minimum width of the stope slice ($A$) should be less than or equal to the width of the motion ellipsoid, i.e.:  
\begin{equation} A \leq 2 \ r \end{equation}

If we consider the value of $r$ as equivalent to that in Equation 6 then:  
\begin{equation} A \leq 4.9 \times S_i \times (1-\varepsilon^2)^{1/2} \end{equation} \hspace{1cm} (Eq. 8)

The optimum design pattern will be the one where the flow zones from adjacent ellipsoids do not intersect but come close enough to provide a significant volume of ore to be extracted from the stope. If the stope slice width is to be defined as per Equation 8, then the pillar width ($P$) (see also Fig. 6) could be simply defined as:  
\begin{equation} P = A - B \end{equation} \hspace{1cm} (Eq. 9)
The main consideration with the pillar width concerns ground stability in that particular area of the mine. The ‘honey-comb’ effect of the combined extraction drifts must also be considered since stress induced instability must be guarded against, particularly in the lower working areas. Ideally, the width of the extraction drift should fit the size of the slice width such that \( B \approx A \). This however would be impossible in sublevel cave operations since the pillar width would then be reduced to zero. A design termed the ‘Silo Method’ has been attempted in Swedish mines where the blasted stope slice is the same width as the extraction drift. This method greatly increases the amount of lateral development per level.

2.3.1 Effective Design of the Extraction Drift

Janelid and Kvapil (1966) have developed an equation to approximate the value for the width of the extraction drift (B) to be determined as:

\[
B > 11.2 \times (ps)^{1/2} \times y \quad \text{(Eq. 10)}
\]

where \( ps \) is a factor developed by Janelid and Kvapil which relates to the size distribution of the fragmented ore and typically ranges between 0.5 and 1.5. The value of \( y \) represents the maximum particle size within that same distribution of rock fragments. The effective width is a function of the shape of the extraction heading and the size fraction of the blasted ore. Developing wide extraction drifts offers two principal advantages:

1. A greater proportion of the ore column’s width is set into motion.

2. The ore-waste interface near the cap remains flatter, preventing premature cap waste dilution into the ore column.

However, ground and span stability, as well as lateral development costs, become considerations to limit design use of overly wide extraction tunnels. As previously indicated, the width of the extraction opening produces changes to the gravity flow of the ore column. In a sublevel cave operation, the opening actually acts as a slot as seen in Figure 7. This creates a strong central flow zone (C) with secondary movement occurring beside this zone (M) in an ellipsoidal fashion. In this situation, mass flow is really only occurring in the central section (C) while the remaining zones of the motion ellipsoid (M) undergo transferred gravity flow.
The width of the extraction drift has a direct bearing upon the width of the mass flow zone as illustrated in Figure 8. Increasing the width of the mass flow zone will increase the volume of un-diluted ore that can be recovered from the ore column. The wider mass flow zone will also be less prone to arching and possible hang-ups since a greater amount of the ore column is in motion at any given time. The wider mass flow zone will be of particular importance in situations where sublevel intervals are to be increased. The reason behind this is due to the high eccentricity of the motion ellipsoids within the high sublevel stopes. The high eccentricity will not allow for significant development of the width of the motion ellipsoids or the limit ellipsoids. This in turn suggests that most of the recovery from these stopes must be from the central or mass flow section of the draw ellipsoid. In effect, the large sublevel intervals result in highly eccentric extraction zone ellipses which require extraction drift widths to be equivalent to the width of the motion ellipsoid. In turn this requires reduced pillar widths and the mining method becomes similar to the 'Silo Method' which will be more fully described in Section 2.4.4.
The shape of the back of the extraction drift has an effect on the width of the mass flow zone. Optimum conditions for improved mass flow require the back to be flat while rounded backs tend to constrict that same flow zone. The relationship between back shapes has been discussed by Kvapil (1992); Figure 9(a), illustrates the narrow cone shaped flow that can be expected from rounded backs. The small effective size of the opening will not enable recovery of the ore in the lower part of the motion ellipsoid and waste entrainment from the cap will ensue before a high percentage of the ore from the column can be recovered. Should the back be flattened, as in Figure 9(b), then the width of the mass flow zone is increased and recovery improved. The ore on the extraction drift floor forms a prism which encourages a wider mass flow zone and maintains a relatively flat contact between the ore and the waste in the descending ore column.
From a ground control perspective, rounded backs are preferred to flat backs in order to alleviate ground instabilities. A balance between the objectives must be achieved in order to decide upon the appropriate roundness of the extraction drift back. A tighter rockbolt pattern, shotcrete and cablebolts, or a combination thereof, would be required to maintain the desired flat back. This solution will add to the development costs and the projected improved recovery must be weighed against the additional support costs.
2.3.2 Draw Rates of the Extraction Ellipsoids

In order to be able to confidently control the discharge of the rock from the sublevel caved stope, it is necessary to understand the effects of draw rates on the ellipsoid of motion. As discharge occurs, the particle velocity of the discharging material is greatest at the extraction opening and along the central axis of gravity flow. Moving both vertically and horizontally away from the outlet decreases the particle flow velocity. This flow velocity will ultimately reach zero along the boundary of the limit ellipsoid. This feature is represented schematically in Figure 10. All velocity vectors in the limit ellipsoid point down and toward the centre of the outlet. In practice, if the outlet width is large enough and uneven drawing occurs, the central axis of the ellipsoids will shift in the direction of the greatest draw. The flow velocity vectors will only develop to the same depth as that of the physical pull. Shallow digging depths will only recover the ore along the contact of the unblasted rock face while the ore at the back of the ring remains stationary.

![Figure 10 - Particle Velocity Characteristics](image-url)
In the sublevel caving system, the ellipsoid of motion is cut off by the extraction drift floor and prevented from developing fully and symmetrically along its vertical axis (Figure 11). The centre line of the ellipsoids will tend to deviate from the vertical by an angle $\alpha$ due to friction (roughness of the wall surface) along the wall. As the wall roughness increases, the angle of deviation will tend to increase.

Figure 11 – Motion Ellipsoid and Limit Ellipsoid Shape with Bunker Style Discharge
2.4 Design Considerations in the Sublevel Cave Mine

One of the most crucial components to successful sublevel caving is proper fragmentation of the ore column. Janelid (1968) reinforces this view when he states that, "The fundamental principle for planning sublevel caving is the knowledge of the gravity flow of blasted rock." and that this, "...exerts the main influence on recovery.". The more finely fragmented ore would tend to have more mobility in the stope area. Accurate drilling and explosive distribution and initiation are critical for uniform fragmentation of the ore column. Thorough fragmentation of the ore column allows for drawing of the ore over the entire width of the extraction drift and for drawing deeply into the muckpile. Both of these factors allow for uniform gravity flow and this promotes a higher recovery of ore and overall effective use of the sublevel cave mining method.

The parameters which affect the design of the sublevel caving mining layout can be divided into 3 categories: geometrical parameters, material flow properties and operational parameters. Geometrical parameters describe the dimensions of the mine layout. Material flow property parameters describe the caveability of the rock. Operational parameters involve the regular work operations at the mine site and are usually controlled by labour and management personnel. Table 1 lists the most important design principles of the aforementioned parameters.

<table>
<thead>
<tr>
<th>Geometrical</th>
<th>Material Flow</th>
<th>Operational</th>
</tr>
</thead>
<tbody>
<tr>
<td>Heading Width</td>
<td>Draw Ellipsoid Eccentricity</td>
<td>Degree of Fragmentation</td>
</tr>
<tr>
<td>Heading Height</td>
<td>Deviation of Ellipsoid Axis</td>
<td>Method of Extraction</td>
</tr>
<tr>
<td>Sublevel Interval</td>
<td>Draw Opening Width</td>
<td>Rate of Extraction</td>
</tr>
<tr>
<td>Draw Height</td>
<td>Rock Densities</td>
<td>Drill Hole Accuracy</td>
</tr>
<tr>
<td>Pillar Width</td>
<td>Internal Friction Angle (ore)</td>
<td>Explosive Loading</td>
</tr>
<tr>
<td>Ring Gradient</td>
<td>Internal Friction Angle (rock)</td>
<td>Explosive Initiation</td>
</tr>
<tr>
<td>Ring Burden</td>
<td>Swell Factors</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Caveability of Blasted Ore</td>
<td></td>
</tr>
</tbody>
</table>

The design of the physical mine layout is essentially a problem of geometrical planning. The dimensions of the layout are invariably dependent on one another and the optimal pattern is one which balances the objectives of the mine plan while bearing in mind the consequences of ore loss and waste dilution. Key geometrical parameters and their impact upon design are outlined below:
2.4.1 Heading Width & Height

The drawpoint opening represents the only immediate free face for the explosive charges. Therefore, a fair amount of the bottom portion of the ring is thrown into this void area. This bottom part of the ring is usually the only section where typical fragmentation occurs since the confinement of the ore on the other sides of the ring does not present void spaces to which the muck may be thrown. The large dimension drifts (5 m x 7 m) may hold approximately 20% of the ring volume from the 30 m sublevel intervals.

The extraction heading height should be kept to the minimum required to accommodate the machinery. The reason for this is to control the bulge that will occur in the draw ellipsoid should this dimension be too great. The ellipsoid bulge will draw material from the back part of the ring which will increase the amount of dilution entering the flow zone. The greater the heading height, the more pronounced the bulge. This draws waste from the ring sooner and to a greater degree than with the lower drift heights. Disturbances from the back of the ring causes ore loss, the extent of which is dependent upon the ring burden, the digging depth of the mucking equipment and the heading height. The deeper digging depths also increase the effective size of the mass flow zone. The importance of these factors is shown in Figure 12.

![Figure 12 - Ore Losses due to poor Digging Depth and Excessive Drawpoint Height](image-url)
The following equation is based upon the Rankin theory and developed by Janelid & Kvapil (1966), which yields \( Q \) as the best digging depth possible by the mucking machinery given the heading height;

\[
Q = H \cot \theta - H \tan (90 - \theta) \quad (\text{Eq. 11})
\]

where; \( \theta \) = angle of repose of the ore
\( H \) = extraction drift height

The Rankin theory is based upon the assumption that the principal stress trajectory of the flowing particles is neither parallel, perpendicular nor vertical to the slope of inclination of the muckpile. The optimum stress trajectory is inclined away from the vertical at a fixed angle. This makes practical sense since, from observation, muckpiles seem to roll as the recovery operation takes place.

The benefits of the lower heading heights must also be balanced with the benefits of providing the blasted ring with an immediate void opening. The effectiveness of the blast and direct ring recovery are dependent upon the area of this void.

2.4.2 Sublevel Interval

The current trend is to increase sublevel intervals by 50 to 100 percent of existing designs. The goal in this is to cut down on the amount of development required to mine a stope block. The increased sublevels require longer blast holes which are more likely to result in inaccurate drilling. However, the advent of improved drilling equipment, which is capable of drilling long and accurate holes, has made it possible for many mines to develop larger stoping areas. For example, large scale sublevel caving has been applied at LKAB's Malmberget Mine, where development was reduced by 50% and stope tonnage increased by over two and a half times. This was accomplished by increasing the sublevel interval by 5 metres (Hustrulid, A., 1995).

The increased sublevels place a great deal of strain on the operation of loading and initiating the explosive column. At Stobie Mine, the explosive column can be as long as 30 m, the toes of these holes are up to 43 m away from the collar. It is difficult to ensure full column loading and bonding at these distances. Also, firing of these columns depends upon collar primed initiators which may not ignite the entire column. Consider that any intrusion of rock or
any air gap of up to 2 to 3 times the diameter of the charge will prevent initiation past that obstruction (Laliberte, M., 1995).

2.4.3 Draw Height

The draw height should be just less than twice the sublevel interval. Draw heights greater than this will tend to draw in waste from the cap rock while lower draw heights will reduce recovery. The eccentricity of the draw ellipsoid and the extraction heading spacing are also both significant factors in controlling ore losses and waste dilution. In the case of the 30 m sublevels, a general conclusion is that pillar width must be increased proportionally to the draw height to accommodate the larger draw ellipsoids. Just (1972) believes that the minimum draw height should be greater than twice the sublevel interval minus the extraction heading height.

2.4.4 Pillar Width

The pillar width relates to the distance between adjacent extraction headings and ideally should be the same as the width of the draw ellipsoid. The width of the draw ellipsoid is dependent on the extraction heading width and the eccentricity of the draw ellipsoid. The optimal spacing should be chosen based on the eccentricity of the draw ellipsoid for that ring. This eccentricity is dependent upon the degree of rock breakage, flow characteristics of the rock and stress regime. Determining the eccentricity is important and even slight changes to its value can seriously alter the draw ellipsoid width. Consider the following; if the draw height is 55 m and the eccentricity 0.95, then the maximum width of the draw ellipsoid would be 17.2 m. Decreasing the eccentricity by 3% to 0.92 increases the draw ellipsoid’s maximum width by over 30% to 21.6 m. Eccentricity values between 0.92 and 0.96 are the most widely accepted for sublevel cave stopes (Just, 1972).

Just (1972) described a hybrid mining method of sublevel cave termed the ‘Silo Method’. The design of this method sets the pillar width equal to the heading width. This configuration allows for vertical channelling of the ore and close to 100% recovery. This method would facilitate drilling and loading difficulties but is impractical due to the increased amount of lateral development work required on each level. The slenderness of the pillars also makes it inapplicable in regions of weak ground or high vertical stress, as was the case at the Mount Isa operations (Chatterjee et. al., 1979). In this case, the limiting pillar width was dictated by the
weak ground conditions. Optimum pillar width requires a balance between optimum ore recovery and pillar stability while limiting the amount of lateral development.

2.4.5 Ring Gradient

The principal reason for angling the ring towards the caved stope is to maintain brow integrity so that charging and initiating of the next ring can be performed safely. Typical ring gradients range from 70 - 90 degrees opposite to the direction on mining retreat. Optimum ring gradient is dependent upon the relative size distributions and densities between the ore and waste rock. The findings of Janelid and Kvapil (1966) are best described in Table 2, where $S_O$ and $S_W$ are the average size of the ore and waste particles respectively.

Table 2 - Rock Size Ratio to Ring Gradient

<table>
<thead>
<tr>
<th>Rock Size Ratio</th>
<th>Ring Gradient, $\phi$</th>
</tr>
</thead>
<tbody>
<tr>
<td>$S_O/S_W &gt; 1$</td>
<td>Forward, $\phi &lt; 90$</td>
</tr>
<tr>
<td>$S_O/S_W = 1$</td>
<td>Vertical, $\phi = 90$</td>
</tr>
<tr>
<td>$S_O/S_W &lt; 1$</td>
<td>Backward, $\phi &gt; 90$</td>
</tr>
</tbody>
</table>

If the ore is coarser than the waste, then the rings may need to be tilted as much as 20 degrees forward. Conversely, an ore to waste rock size ratio much less than 1 may require a 20 degree backward tilt. The backward tilting ring will, theoretically, prevent the finer blasted ore from being lost in the void space of the caved waste rock. This idea is meritorious but the benefits of regaining the finely blasted ore becomes insignificant when weighed against the chance of losing the brow and leaving behind an intact ring. In summary, backward tilting has two major effects; first, it reduces the efficiency of production drilling and second, it greatly reduces brow control and stability. Forward tilting will continue to be the normal course of operations and this will only change in circumstances where brow integrity is guaranteed not to fail. However, even forward tilting rings tend to erode brow stability and competence.
2.4.6 Ring Burden

The ring burden is the horizontal distance between blast holes in adjacent rings. The ring burden, or simply ‘burden’, is one of the most flexible parameters in the sublevel cave layout since it can be altered at various planning stages. Other important features include:

- The burden can be directly changed even between adjacent rings.
- Burden size has immediate implications on the recovery capability of the flow ellipsoid.
- Burden size may be limited by blasting operations.

If ellipsoidal draw shapes are to be expected from the sublevel stope, then the maximum ring burden would be half the maximum width of the draw ellipsoid. This is also referred to as the semi-minor axis of the ellipsoid (b_n). A burden of this size will reduce the likelihood of drawing in dilution from the wall waste region. Minimum burden size has been arbitrarily set to about a quarter of the draw ellipsoid width.

Excessively large burdens cause ore loss since none of the ore beyond the limit ellipsoid is directly recovered. It has been postulated that this ‘wedge’ of ore may be recovered on subsequent levels. However, the writer believes that this recovery would be impossible to depend upon especially when considering that it must flow a distance of twice the sublevel interval to reach the drawpoint. Thin burdens cause dilution since the extraction ellipsoid digs into the caved waste from the previously fired ring. The best value lies between these limits where a satisfactory compromise between recovery and dilution can be reached.
2.5 **Recovery and Dilution**

A successful sublevel caving design is one which optimizes recovery and dilution. These two factors are, potentially, the two greatest weaknesses of the method and improvement in these areas would result in a more efficient mining method. Recovery can be defined as the ratio of the number of tonnes of ore drawn versus the total amount of ore blasted. Dilution is defined as the ratio of tonnes of waste drawn to the number of tonnes of ore and waste rock. Extraction is the total amount of waste and ore drawn divided by the tonnes of ore blasted in the ring. These parameters can be defined mathematically as:

- **Recovery**: \( R \) = \( \frac{TW}{TB} \) tonnes of waste drawn
- **Dilution**: \( D \) = \( \frac{TW}{TD} \) tonnes of ore blasted
- **Extraction**: \( EX \) = \( \frac{TO}{TB} \) tonnes of ore drawn
- **Total**: \( TD = \) tonnes of ore + tonnes of waste drawn

Percentage expressions can be defined as follows:

- **Recovery**: \( R = \frac{TO}{TB} \times 100 \)
- **Dilution**: \( D = \frac{TW}{TD} \times 100 \)
- **Extraction**: \( EX = \frac{TD}{TB} \times 100 \)

Combining the equations for Recovery and Extraction also yields:

- **Recovery**: \( R = EX \left( 1 - \frac{D}{100} \right) \)

In practice, recovery varies between 60% and 90%, while dilution varies between 5% and 40%.
Extraction efficiency (E), is perhaps a more useful measuring tool for determining the recovery/dilution characteristics of a sublevel cave stope. This ratio can be described as:

\[ E = R \left(1 - \frac{D}{100}\right) \]

This equation indicates that if dilution is low and recovery is high, then the extraction efficiency is high. Although dilution and recovery are affected by blasting efficiency, mucking techniques and draw control, the basic mine layout has impact upon these functions and may be controlled by design engineers. The designer does not have direct control over the material flow properties and fragmentation of the rock column. Mine personnel are able to control only the geometrical and operational parameters of the sublevel cave stope. However, both of these factors have significant impacts upon the material flow properties of waste and rock.

The motivation for minimizing dilution is critical to the sublevel cave operation for three reasons:

1. The blasted ring is usually bound by waste on three sides; at the back, to the top and on one of the sides of the ring. This provides 3 openings to dilution.

2. Mucking overlap at the drawpoint will cause non-uniform draw which may also disturb the uniformity of the ore column.

3. The individual rings are judged on their recovery and dilution ratios. The decision to continue mucking a particular ring is largely dependent upon the mine’s cut-off grade and the dilution encountered at the drawpoint.

Draw control is most commonly done by visual observation, particularly in cases where the ore and waste are different in color. Other methods, such as magnetometer readings and probe devices have also been useful in grading the muck at the drawpoint where the difference between ore and waste is discernible due to magnetism or specific density. In large scale sublevel cave, the beneficiation plant must be able to provide for the increased through-put as well as the lower grades that are to be expected from the method.
3.0 PHYSICAL MODEL TESTS

The main purposes of using a physical model in this study were to:

- provide an understanding for the flow of bulk solids in a sublevel cave environment
- determine the effects of altering geometrical parameters to the sublevel cave layout
- predict recovery and dilution rates based upon model design
- determine the validity of using scaled models to forecast the flow of ore in a stope

Fundamentally, the model gives an indication of the ellipsoidal flow motion associated with the sublevel cave stope. This model was also used to study the effects of scaled particle sizes within a heterogeneous mixture. This is considered a significant point since most physical models have usually observed the flow of sand, a homogeneous material the particle size of which had no relationship to the size of the container. Thirty-two model tests were conducted from April to July 1995.

3.1 Model Description

3.1.1 Model Profile and Construction Material

The physical model used for the study were a 1 : 48 scaled representation of a typical sublevel cave stope block. The physical dimensions of the model were 83 cm wide by 79 cm deep by 114 cm high. Three extraction drawpoints were placed along the base allowing for simulated mucking to the rock column. The model is illustrated in Figure 13. The containing walls of the model were composed of 0.64 cm plexi-glass held in place by steel angle irons at the corners. The model was further reinforced by 0.95 cm steel rods which help the model to remain rigid on loading. The capacity of the model was approximately 300 kg of rock which was loaded at the top and recovered from the base. A long piano hinge was located just above the drawpoints so that the plexi-glass face of the model could be tilted to simulate various ring
angles during test runs. After test #3, a 1.27 cm square wire mesh was installed along the interior of the model face to simulate the expected increased friction coefficient along this surface.

![Diagram of Physical Model](image)

Physical Model Dimensions
- Height: 114 cm
- Width: 83 cm
- Depth: 79 cm

Adjustable Face
- angles 70 to 85°

31 m Level

21 m Level

White Quartz Ore Column
- Burdens set to 2.74 m, 4.12 m & 5.49 m

Figure 13 – Physical Model

3.1.2 Model Design

The model was designed so that various geometrical patterns could be tested and analyzed. Figure 14 shows a photo image of the actual model in an unfilled state.
The geometrical parameters studied include:

i) Sublevel Interval (21.34 m and 30.48 m)

ii) Ring Spacing (2.74 m, 4.12 m and 5.49 m)

iii) Ring Gradient (75°, 80° and 85°)

iv) Drift Width (4.42 m and 5.49 m)

v) Pillar Width (12.19 m & 15.24 m)
Remarks:
i) The sublevel intervals of 21.34 m and 30.48 m were measured from the floor level of the lower extraction cross-cut.

ii) The ring spacing was controlled by a piece of plexi-glass fixed in place for the appropriate burden size. The white quartz representing the ore for these tests was then poured into place between this divider and the adjustable model face.

iii) The ring gradient was set by the adjustable hinged face while the plexi-glass divider used for ring spacing was set parallel to this angle.

iv) The drift width could be changed since the plexi-glass drawpoint set up below the hinge on the model face could be entirely removed and replaced.

v) Similarly to iv), the cross-cut spacing could be altered.

In addition to the dividers used to measure ring spacing, dividers were also set between the individual stopes. This was done in order to compartmentalize each stope block so that accurate weights of rock within each area could be determined.

3.1.3 Model Materials

Three types of crushed rock were used in the model.

Ore: The ore in the tests was represented by white quartz. The size fraction was blended into a heterogeneous mixture which was + 0.64 cm to − 1.27 cm in size. This size fraction represents ore pieces which range in size from 30 cm to 60 cm in the actual mine setting. The specific density of the white quartz was 2.7 g/cm³.

Wall Waste: The waste which is found in either the hangingwall or footwall in the mine environment was represented by a pink quartz evenly blended to a mixture comprised of the following size fractions; 33% (0.64 cm to 1.27 cm); 33% (1.27 cm to 2.54 cm); and 33% (2.54 cm to 4.45 cm). The size distribution was an attempt to closely simulate actual underground experience with respect to wall waste size fractions. The size distributions for both types of waste particles were modelled after the grading curve shown in Figure 15. This distribution
acknowledges the larger size fraction expected from the caved wall rock which is not subjected
to direct blasting. During the experimental trials, the pink quartz was placed directly behind the
ore column. The representative pink quartz had a specific density of 2.7 g/cm³, identical to the
white quartz.

**Cap Rock:** The cap waste in the model was represented by black gabbro blended to a
heterogeneous mixture identical in size distribution to the wall waste. This was done to emulate
the larger size fraction expected from the cap rock while accounting for the possible self
comminution effect as this top rock percolated a long distance down to the drawpoint. The black
gabbro had a specific density of 2.8 g/cm³.

The crushed rock used in this project was obtained from Clarabelle Mill in Copper Cliff
and was chosen primarily because of its visual distinctiveness and uniformity of specific
densities.

![Waste Grading Curve](image)

**Figure 15 - Waste Grading Curve**
3.1.4 Material Shape and Uniformity

Particle Shape

The shape of rock fragments is important in determining their kinematic behaviour. Because particle shape is difficult to quantify it is generally disregarded. Qualitative terms for particle shape are most useful and are defined as: bulky, flaky or needle-like. Only the bulky particle shape will be discussed here. Bulky particles have relatively similar values in all three dimensions and are typically formed from mechanical and blast actions. The particles used in the physical model and in the mine are bulky in nature. The aspects of sphericity and angularity are of particular importance to bulky particles.

The sphericity value describes the difference between the length (L), width and thickness. The equivalent diameter of the particle is the diameter of the sphere of equal volume (V₀). The particle’s equivalent diameter (Dₑ) is mathematically defined as:

\[ Dₑ = \sqrt{\frac{6 \, V₀}{\pi}} \]

From the Dₑ, the sphericity (S) may now be defined as:

\[ S = \frac{Dₑ}{L} \]

A perfect sphere has a sphericity of 1. Values lower than this indicate elongated particles. The fragments used in the physical models had values of S = 0.84 to 0.96, indicating largely spherical particles.

The angularity of roundness (\( \angle R \)) is generally a measure of the length of the particle’s asperities and is quantitatively defined as:

\[ \angle R = \frac{\text{average radius of corners and edges}}{\text{radius of maximum inscribed sphere}} \]

Because of complications in its measurement, particle angularity is usually qualitatively described as indicated in Figure 16. The particles used in the physical model are best
characterized as subangular to subrounded. The mechanical fragmentation caused by blasting tends to create angular particles. In addition, the caving action in the stope wears down the sharpness of the asperities to produce a more rounded specimen.

![Particle Angularity](image)

**Figure 16 – Particle Angularity**

**Particle Uniformity**

The effective size \((D_{10})\) of particles is defined in a sieve analysis where 10% of the batch passes through a particular sieve size. The uniformity coefficient \((C_u)\) is defined as:

\[
C_u = \frac{D_{60}}{D_{10}}
\]

which is a ratio between the sieve size where 60% of the material passes through to that of the \(D_{10}\) size. Uniform particles have \(C_u\) values of 4 or less. Both classes of particles used in the physical models can be described as uniform. The \(C_u\) for the ore was less than that of the waste material. \(D_{60}\) for the waste was 2.25 cm and \(D_{10}\) of the waste was 0.79 cm (Fig. 15). The low uniformity coefficient for the ore reflects the assumption that it is a uniformly blasted column for the modeling process.
3.2 **Test Procedures**

3.2.1 **Loading**

The model was loaded as follows:

i) The dividers and geometrical parameters within the model were set to their intended positions and the weights of the respective rock types were determined. An electronic scale, accurate to one one-hundredth of a pound, was used to weigh. It should be noted that the weight of the upper blasted apexes are included in the weight of the ore column since these areas are deemed as recoverable.

ii) Using 5 gallon pails, the wall waste at the back of the ring was poured into place behind the plexi-glass divider which separated the ore and rock columns.

iii) The ore column was brought to a representative height of 9 m and subsequently layered in 6 m intervals until the apex height was achieved. This step was taken so that coloured markers could be placed at the various levels in the ore column to attempt to determine the movement of the ellipsoidal flow in the third dimension. The markers were placed by using a 1.27 cm copper pipe, located equi-distant along the ore column’s length.

iv) Funnels and tubes were used to construct the triangular shaped apex which represents the top section of the ore column. In reality, this section is the pillar section between two production tunnels located 2 sublevels above the current drawpoint. This angle varied between 70° and 80° since it was dependent upon sublevel interval and drawpoint width. In order to maintain uniformity for the model, the upper apex was set at 75° for all test runs except for the last two trials (tests #31 & #32). These trials had upper apex angles of 80° with 5.49 m wide drawpoints.

v) The apex was bound on all sides by cap rock which was poured into the remaining top section.

vi) During the loading process, the model was periodically tapped by a rubber mallet so that the particles could settle.

vii) After loading was complete, a weight of 68 kg was evenly distributed over the top of the cap rock to simulate overburden stress. Sand bags were placed on the model to represent this weight.
3.2.2 Drawing

The rock was drawn from the drawpoints by means of a representative scoop bucket. This bucket was scaled to match the volume and digging depth that would be expected from a 6 cubic metre Scooptram (or 8 yard LHD). The drawpoints were drawn (mucked) simultaneously with 10 buckets withdrawn from each heading in sequential order. A limited number of tests were completed wherein one drawpoint was mucked in its entirety until continuing on to the next drawpoint. Mucking continued until dilution appeared in the ore column. This amount of withdrawn rock was weighed and noted as being the undiluted portion of the ore recovery. After the undiluted ore was removed, the source and weight of the waste dilution was noted. Observation of the waste gave an indication of the development of the flow ellipsoid.

The weights of the ore, wall waste and cap rock were recorded so that recovery and dilution percentages could be determined. Markers within the ore column were also recovered and the scoop bucket count at which they entered the drawpoint was noted. This provided an indication to the relative vector velocities, eccentricity and form of the draw ellipsoid within the ore column. Bucket counts for 30% and 50% dilution marks were also recorded since these represented the mine and model cut-off points respectively.

3.2.3 Unloading

When the 50% dilution point was achieved, the model face could then be removed so that the remaining rock in the model could be manually shoveled out. Efforts were made to maintain the unmixed sections of the rock columns in order to reduce the amount of manual sorting required per test run. The best developed method for this task was to insert plexi-glass dividers between the various rock types. It was difficult to place the dividers since the weight and total friction of the rock column was not easily overcome.
3.3 Model Assumptions

In order to simplify the model testing procedure various assumptions about the material and model parameters were made, making evaluation of the results possible. The seven assumptions follow:

1. **Friction**: The plexi-glass sidewalls of the model were considered smooth and non-restrictive to the motion of flow of the rock material while the friction coefficient of the front wall was increased by means of a 1.27 cm wire mesh. This point was considered valid primarily due to the smoothness of plexi-glass. Also, the interaction of the sidewalls to the flow ellipsoid would only occur on drawpoints #1 and #3 (those stopes at either end of the model). Further, the zone of this interaction was limited to the region where the draw ellipsoid was widest. This typically occurred at a section found at approximately two-thirds of the height of the limit ellipsoid and, since the widening of the motion ellipsoid to this limit occurred gradually, this interaction took place for a limited time near the end of the drawing cycle. The wire mesh on the front face of the model attempted to simulate the increased friction expected from the unblasted rock face. This frictional value was probably understated in the model since in the actual mine environment, this surface may be jagged and undulating. However, the wire mesh provided a better option and more realistic results to leaving this surface smooth since a smooth surface would increase the likelihood of flow along this front face.

2. **Particle Size & Distribution**: The rock size distribution, particularly in the ore column, was uniform in both size and shape. This assumed near perfect blasting techniques since the ore column was uniformly fragmented along its entire length. In the mine environment the objective is to achieve uniform rock size, although this is not realistic since problems associated with loading, charge initiation and drill hole alignment make it difficult to ensure full fragmentation of the ore column, particularly toward the toe of the drill hole. However, in order to maintain a standard for the model test trials, the particles were uniform in size and shape. This standardized the tests. It also reflected the fact that particle size distributions
could not be predicted with great accuracy on a regular basis even though larger-sized particles could be expected towards the ends of the holes.

3. **Particle Angularity:** The crushed rock used in the model was subangular and subrounded for all of the model trials. It was further observed that these traits changed minimally during the course of the testing procedure. Comminution during draw operations did not occur. This was attributed to the reduced scale and the relatively high strength of the particles within the model. The smaller scale did not allow for high gravitational forces to develop; these are required for true comminution to take place. Also, the strength of the particles was also so high that allowing them to free-fall from the top to the bottom of the model would not break them. The particles in this small scale environment had relatively high strengths while being small enough so as to not exhibit any planes of weakness. Minor chipping and rounding of the particles did occur during draw procedures. This mainly frictional action tended to knock off the sharp asperities of the subangular particles as the entire particle set became more uniformly subrounded. This action produced fine grained sand and rock dust within the model.

4. **Flow Action:** Direct observation of the model could only account for the two-dimensional flow of the drawing action. During trials, it was only the front face that could be observed; flow patterns showed a strong central core flow. The markers used during the tests attempted to define the flow within the motion ellipsoid but irregular marker recovery made it unreliable to depend upon this data. During each of the test runs, it appeared that the fastest motion was toward the front of the central flow channel. The speed of the middle and back of the flow channel could not be accurately measured and it is assumed that particle velocity decreased towards the back of this central mass flow zone.

5. **Packing:** Loading the model resulted in random packing of the rock particles. The rounder shape of the particles within the ore column made packing more uniform but the elongated shapes of some of the waste rock assured a higher degree of preferential packing*. This point was considered valid since the settling of particles always occurred when the model

---

* Preferential packing indicates a low void area in the particle assemblage.
was vibrated by the tapping of the rubber mallet. This random packing was not considered a deterrent to the model results as packing within the actual mine stope is similarly difficult to control.

6. **Vertical Stress:** The vertical pressure decreased marginally during the draw procedure and was not considered a major disruptive factor during testing. As drawing progressed, rock was not added to the top of the model so as to maintain the same vertical stress. But the 68 kg sandbags remained in contact with the cap waste which settled at the same rate. This method imitated the mine environment. The reduced scale of the model made this assumption practical since the maximum stress differential in the model was 30 kPa, while the maximum actual force differential over the entire ore column was less than 230 Newtons. In comparison, the maximum stress differential in the mine setting would be 1500 kPa, a multiple of 50, reflecting the model's scale.

7. **Horizontal Stress:** Horizontal stress was not considered a factor during test trials. In the model set-up, horizontal stresses were present due to the bowing of the plexi-glass. This was particularly evident in the mid-section of the front face though flexing also occurred at the sides of the model. The 0.95 cm steel rods, mentioned in the materials section, did increase the stiffness of the front face so that it remained relatively flat. The unconsolidated rockfill which bounds the typical sublevel cave stope prevents significant levels of horizontal stress. This caved fill further reinforces the idea that horizontal stresses are not a significant factor in the model or in the mine.
3.4 Data Collection & Valuation

3.4.1 Data Sources

The test runs involved collecting values for recovery and dilution as described in the Test Procedures section. These values allowed for quantitative comparisons between the various sublevel cave geometrical configurations and were obtained by sorting and weighing the various classes of rock. Computer images were captured with an ‘Apple Quicktake’ bitmap imaging camera; this provided still pictures of the progression of the rock flow. These images were taken after at interval of 10 scoops, providing a total of 560 pictures. Test #26 was recorded to completion with an 8 mm video camera allowing better observation of the flow of the rock in the physical model. The video also showed the entire process of the operations such as the weighing and sorting of the rock classes, the bitmap camera set-up as well as data collection. The video effectively captured the settling process that took place when the model was tapped during the unloading process, and showed the degree of interlocking which took place during drawing.

3.4.2 Model Calibration and Modifications

The first six tests were run to calibrate the model and to set the standard method of operation. The model is calibrated to closely imitate the results from the 21.24 m sublevels at Stobie Mine. Although the results were never exact, they did increase the confidence level in the manner in which the model was functioning. Lessons learned about the design of the upper apexes during the first six tests were incorporated into subsequent tests. The primary problem with the apexes was that they settled into the ore column and therefore had to be built differently. This was accomplished in two ways: first by over-stacking the top volume of the apex and, second, by tapping and packing the model during loading. The results from tests #1 and #2 indicated unexpectedly low recovery values, which suggested the importance of recovering ore from the upper apex area.

Results from the first 6 trials also produced particularly low values for the amount of wall waste included in the total dilution. The size fraction for the wall waste was 100% (+ 1.27 cm to
- 2.54 cm), with this size distribution caving very little during drawing. This fact was recognized while emptying the model during the sixth trial. As the rock was removed, a void formed which allowed for observation of the pink wall waste from the front face. Through the void, the wall waste maintained its original angle of placement of 80° and did not inter-mix with the ore column. This is considered contrary to expected underground behaviour. The rock from this area should be allowed to flow should the motion ellipsoid come into contact with it. This did not occur with the large sized wall waste, which tended to inter-lock and form a wall, preventing the pink wall waste from entering the flow ellipsoid. Very small chips of wall waste getting through to the drawpoint provided the clue that the ellipsoid was reaching the draw area. However, the ellipsoid could only pull the chips which were free to move from unconfined spaces within the wall of pink waste. Following the sixth trial, the new wall waste size distribution was used as described in section 4.1.3 Model Materials, and the wall waste became more apparent in the tests which followed. The effect this had upon dilution is that wall waste dilution accounted for an average of 66% of total dilution as compared to the previous average of 12% for identical test configurations.

In the underground environment it is difficult to ascertain whether the waste is from the wall or cap rock. However, due to the 'freshness' of the waste that enters the drawpoint, indications are that a fair amount of the waste must be from the wall rock, although the exact percentages are debatable. Underground observation makes it reasonable to expect over half of the waste to be from the wall rock, hence an average value of over 60% is acceptable. Also, the size of the waste from the wall rock in the actual mine indicates that the ellipsoid does recover rock from this area. This further substantiates the move to a more mobile size distribution for the wall waste area since large sized waste particles, and not just chips or flakes, are entering the drawpoint (see Section 4).

The combination of the modifications listed above increased recoveries by over 11% to bring recovery values up to levels which were more expected. These changes improved the functioning of the model in two ways:

i) Physical model results more closely matched the values found in the underground environment.

* The freshness indicates that the rock is recently blasted and un-oxidized. Cap rock tends to be highly oxidized and large in size.
ii) The model behaviour, particularly the flow of waste particles, improved to become more realistic.

3.4.3 Data Evaluation

The most significant values for the tests were the recovery and dilution values. These were calculated at intervals of every 10 to 20 scoops until the 50% dilution limit was reached. The basis for recording these values was on percent weight, so result accuracy was very high with little to no chance of random error. This is in contrast to the method used in the actual mine site where dilution is calculated visually at the drawpoint and representative values are determined by the geologists for ore and waste. The percent weight method was adopted for the physical model mainly to reduce the large percentage of errors that could occur given the reduced scale of the model.

Recovery

The value for recovery was determined by first separating the white quartz from the pulled material and subsequently weighing that amount of quartz. The total amount of ore or white quartz in the area of the ideal ring was weighed and recorded prior to loading the model. The area for the ideal ring is shown in Figure 17, where the hatched region represents the recoverable ore for that particular ring. From this diagram it is interesting to note the apex angles at the lower and upper sublevels. The lower angle represents the side hole angle used in the plan while the upper angle is equal to the internal cave angle. The internal cave angle is the angle formed by the draw ellipsoid with the horizontal at the side of the extraction heading.
Figure 17 – Recoverable Ore Ring
Dilution

The value for dilution for both the model tests and at INCO’s Stobie Mine is calculated as an instantaneous dilution rate:

\[
\text{% Inst. Dil.} = \left( \frac{\text{mass of waste}}{\text{mass of ore}} \right) \times 100
\]

Instantaneous dilution is a measurement of dilution of the drawpoint at an exact period of time and is obtained with the formula noted above. This method is analogous to taking a still photograph of the drawpoint and determining its dilution over a specified interval.

\[
\text{%Ov. Dil.} = \left( \frac{\text{mass of waste}}{\text{mass of waste} + \text{mass of ore}} \right) \times 100
\]

Overall dilution rates which were calculated with the formula above were also determined for the tests. Overall dilution is a true percentage evaluation of the amount of waste found in the mucked material and is the most popular method employed by most other studies involving dilution (Section 2.5).

Instantaneous dilution and subsequent recovery values were noted at the 30% and 50% limits. Due to the sensitivity to dilution at Stobie Mine, the mine personnel consider 30% instantaneous dilution as the end point of the mucking cycle for that ring. It is significant to note that while instantaneous dilution reached 50% in the models, the value for overall dilution on 75% of the tests conducted was still below 10%. This raises the question that dilution restrictions may be set too conservatively. This question becomes particularly valid when considering that other bulk mining methods typically have overall dilution rates in the range of ten to twenty percent.

3.4.4 Data Statistics

As previously noted, the first six tests were inadequate leaving twenty-six tests useful for data purposes. The first twenty-three tests had cross-cut dimensions scaled to represent a 4.27 m
high by 4.42 m wide drawpoint with drawpoint spacing set at 12.19 m. The last 3 tests had cross-cut dimensions of 4.27 m high by 5.49 m wide with 15.24 m drawpoint spacings, and these tests were only performed for the 30.48 m sublevel intervals. For those tests with the wider drawpoints, the physical model could only accommodate 2 cross-cut headings since the pillar’s scaled width was 9.75 m, and the model simply did not have the length to fit the extra extraction heading. Table 3 provides a breakdown of the number of tests conducted per geometrical configuration. This analysis considers only the 4.27 m high by 4.42 m wide drawpoints with 12.19 m drawpoint spacing for comparison.

Table 3 - Test Statistics

<table>
<thead>
<tr>
<th>Sublevel Interval</th>
<th>Ring Burden</th>
<th>Ring Gradient</th>
<th># of Tests</th>
<th>Average Recovery*</th>
</tr>
</thead>
<tbody>
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<td>2.74 m</td>
<td>2.74 m</td>
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<td></td>
<td></td>
<td>80°</td>
<td>4</td>
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</tr>
<tr>
<td></td>
<td></td>
<td>85°</td>
<td>0</td>
<td></td>
</tr>
<tr>
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<td>4.12 m</td>
<td>75°</td>
<td>1</td>
<td>61.6</td>
</tr>
<tr>
<td></td>
<td></td>
<td>80°</td>
<td>1</td>
<td>69.6</td>
</tr>
<tr>
<td></td>
<td></td>
<td>85°</td>
<td>0</td>
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</tr>
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<td>75°</td>
<td>1</td>
<td>68.3</td>
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<td>71.2</td>
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</table>

* Recovery values were determined when instantaneous dilution reached 30%. Overall dilution at this point was usually less than 5%.

As indicated in Table 3, the highest recoveries for the 21.34 m sublevel interval were achieved with a 4.12 m burden and a ring gradient of 80 degrees. The 30.48 m sublevels had best results when the burden was 5.49 m with a ring angle of 80 degrees.
3.5 Physical Model Results

The results for the physical model tests are compiled in Appendix 1 - Physical Model Results. The appendix contains two tables: Table 1 is ordered such that the results from the model are given in the numerical order in which they were completed, while Table 2 groups the results of similar geometrical configurations. These tables show recovery and instantaneous dilution values as well as overall dilution, internal cave angle, and the dilution source. The dilution source would be from the cap or the wall rock. The internal cave angle may be thought of as the angle of repose of the moving motion ellipsoid. The internal cave angle represents the inclination of the slide on which material within the stope enters the drawpoint. This angle is readily apparent on either side of the drawpoint and it is expected that this slide projects into the ore column at the back of the ring. The action and importance of this angle will be discussed later in this chapter.

3.5.1 Twenty-one metre Sublevel Interval Results with 4.42 m wide Drawpoints

The 21.34 m tests were performed primarily to calibrate the model and to ensure that the model was functioning properly as well as to provide insight into the flow of material within the model. Tests 7, 8, 21 and 27 are considered to be the base trials which best exemplified the actual mine conditions at Stobie Mine.

Figure 18* displays the summary of the recovery values obtained from the various geometrical configurations tested when instantaneous dilution reached 30 percent. The best results for the 21.34 m pulls were achieved when the burden was 4.12 m and the ring angle was 80 degrees. The 2.74 m burdens yielded high results while the 5.49 m burdens posted the lowest results. The ellipsoid of motion does not develop far into the larger burdens and much of the ore in these deep columns remains stationary during the pulling of the model. The 80° ring gradients exhibited higher recoveries than their 75° counterparts for each burden size. The greatest percentage increases, with regard to the steeper ring angles, was evident with larger burdens.

* Note that the base of the graph begins at the 50% recovery so that the differences in recovery values can be accentuated.
3.5.2 Thirty-one metre Sublevel Interval Results with 4.42 m wide Drawpoints

The recovery values at the 30% instantaneous dilution point for the thirty-one metre pulls with 4.42 m wide drawpoints are summarized in Figure 19. The general conclusions drawn from these tests indicate that larger burdens and steeper ring angles provide the highest recoveries.

The optimum configuration in this group was the 5.49 m burden with a ring gradient of 80° while similarly high recoveries were obtained from the 85° rings. The 5.49 m burden inclined at 75° showed lower recovery because the motion ellipsoid did not fully develop into the lower section of the ore column. This point is of significant importance and is similar to the observation made for the larger burdens in the 21 m pulls. It proves that the characteristic of the particles' motion is strongly influenced by the combined effects of the burden and ring gradient.
Figure 19 - 31 m Sublevel Interval Results

Large burdens improve recovery when the ring gradients are steep, as in the case of moving from the 4.12 m burden to the 5.49 m at a ring gradient of 85 degrees. In this case, the recovery increased by approximately 37%, indicating that the 4.12 m burden was too narrow to
accommodate the near vertical angle. Large burdens did not improve recovery for the flatter ring gradients, as indicated by the case with the 5.49 m burden at a 75° ring angle when compared to the 4.12 m burden with the same gradient. In this situation the recovery actually drops by 3% from the 4.12 m burden.

The results from both the 21 m and 31 m tests cannot be expressed in terms of linear relationships wherein recoveries would increase or decrease proportionally in relation to varying either the burden or ring angle. Instead, results seem sensitive to the appropriate combination of these two geometrical factors. These observations indicate that each burden size has a particular optimum ring gradient. This influences the form of the motion ellipsoid which in turn determines the volume of ore within that column which may be recovered upon extraction.

Having an optimum ring angle per burden size also indicates that the traditional ellipsoid of motion does not fully develop within the fragmented ore column. Evidence of this fact can be found in the non-linear nature of the results where altering the ring angle or burden would proportionally increase or decrease its associated recovery value. Therefore, the rounding of the ellipsoid towards the back of the ore column does not occur to any great extent within the physical models. The sum of these observations indicates that the characteristic of particle flow within the physical model does not conform to the traditional motion ellipsoid within the bounds of the ore column.

3.5.3 Thirty-one metre Sublevel Interval Results with 5.49 m wide Drawpoints

Three tests; numbers 30, 31 and 32 were performed with a sublevel interval of 31 m at 15.24 m drawpoint centers with 80° ring angles and 5.49 m wide drawpoints. Of these tests, the results from test 30 were not considered for analysis since the apexes built for this trial were too small to account for the 15.24 m drawpoint spacing and hence showed lower recovery values. However, this test did prove the importance of recovering the ore located in the upper apex, as recovery improved by 10% with the appropriately sized apex given the increased drawpoint spacing. The results from tests 31 and 32 are summarized in Figure 20 which shows the recovery values for the 5.49 m wide drawpoints when 30% instantaneous dilution is attained.

The results from these tests demonstrated the highest recovery values of all the geometrical configurations attempted during the experiment. This indicated that the width of the
central channel, or zone of core mass flow, is particularly important to yielding high recoveries. The core mass flow zone has the highest mobility within the stope and is situated directly above the drawpoint.

Consider that the primary influence of the extraction heading width is that it provided the direct outlet for the ore during the course of pulling from the drawpoint. The wider drawpoints increased recoveries since a greater percentage of the ring's cross-sectional area would be situated directly above the opening and thus subject to motion. The results from all of the physical models showed that the motion above the drawpoint had the highest degree of movement and vector velocity strength. The results from the 5.49 m wide drawpoints proved the importance of increasing the area of the zone of motion to increase recovery as caving progresses to the drawpoint.

The best recoveries in this set-up were obtained from the 5.49 m burdens. This result reinforces the idea that there is a sense of proportionality associated with the results obtained from the physical model. Since the main objective is to increase the sublevel interval, it is reasonable to assume proportional increases to burden and ore slice width; this is indicated from the model's results. Similarly, the 15.24 m drawpoint spacing increased the width of the ore slice and was provided for by the wider drawpoint.
3.6 Dilution Location and Flow

The source of the dilution, whether it is from the wall or the cap waste, indicated the relative flow strength and direction of the particles within the model. This gave an indication as to whether the characteristic of particle movement was behaving in the traditional ellipsoidal manner or if the flow was more vertical in nature. Presumably, the best configurations would have a 50/50 split between wall and cap waste where both waste fronts meet at the same time at the mouth of the drawpoint. If this occurred, then recoveries would be maximized since the ore between these locations would have been forced to the drawpoint.

Figure 21 provides a breakdown of the recovery values and dilution source for the various ring specifications for the 31 m sublevel intervals with 4.42 m wide drawpoints. The recovery values were obtained at the 30% instantaneous dilution point; similar to the case in the previous three graphs. The results are ranked in order, where the bar on the extreme left represents the highest recovery and the bar furthest to the right denotes the lowest.

![Figure 21 - Dilution Source](image)
The dilution source is represented as a percentage of the bar where a completely shaded bar indicates that all of the waste was from the wall rock. The best results were achieved when there was an even proportion of both waste types or at least a high percentage of cap rock within the total dilution. The burdens indicated in the first two bars are 5.49 m, showing the importance of a strong vertical flow for the larger burdens. The 5.49 m burden angled at 75° had a lower recovery with 100% of its dilution originating from the cap rock. This implies that much of the ore located toward the back of the ring was left in place, un-recovered. The high amount of cap rock for this configuration indicates that the strong vertical flow associated with the 31 m sublevel intervals did not move the entire burden at this ring gradient but mobilized only the front part of the ring where flow strength is highest.

The 4.12 m burdens displayed a direct relationship to ring angle where recovery decreased as the ring gradient increased. The highest recovery in this group corresponded to the ring angled at 75° where almost 80% of the dilution was from the cap rock. The 2.74 m burdens behaved similarly to the 4.12 m burdens in that the highest recovery corresponded to the flattest ring gradient; the 75° ring showing signs of cap rock within the dilution. The appearance of cap rock is important to all of these configurations since its inclusion indicated movement along the vertical axis of the ore column. Considering 31 m sublevels, the importance of vertical flow is important.

At this point it is evident that there is a complex relationship developing between geometrical configuration and ore recovery since there does not appear to be a linear relationship between ring gradient or burden size. For example, the results from Figure 20 and its corresponding data Table 2 found in Appendix 2 indicate that the slimmer burdens cannot accommodate the steeper ring gradients. The amount of wall waste for these configurations exerted a large influence on the total amount of dilution, contributing to the fact that the ore along the vertical axis of the ore column was cut-off from being recovered. The 5.49 m burdens provided a different perspective. The optimum recovery for these burdens was at 80° while recovery remained similarly high for the steeper ring gradient of 85°. The lowest recovery value for this burden was at 75° where 100% of the waste was from the cap rock.

Charting the source of the dilution further reinforced the notion that recovery is dependent upon the combination of ring angle and burden size. It has also provided a great deal of insight
into understanding the characteristic of the particle’s motion within the physical model and it is with this understanding that sublevel cave operations can be improved.

3.6.1 Effect of Dilution Source upon Recovery

The source of the dilution within the physical model gave a good indication of the regions of strongest flow. As seen in the previous section, those configurations with the highest recovery values had strong vertical flow with cap rock waste evident in the dilution. A particle flow comparison between the best configuration, Test 17, with a 5.49 m burden angled at 80° and the worst configuration, Test 29, with a 4.12 m burden angled at 85°, is shown in Figures 22 and 23 respectively.

Figure 22 (a) shows the model after 30 scoops had been removed. The ore extracted at this point was without dilution, similar to the model when it was newly loaded. The second image (b) shows the model approximately two-thirds of the way into the test run. At this point, the waste rock in the upper drawpoint areas had dropped significantly. This indicated the strong vertical movement along the entire width of the ore slice of this particular configuration. The waste from the upper drawpoint (DP) area widened considerably as it progressed down toward the extraction drift.

It should be noted that this upper drawpoint waste was actually 30% instantaneously diluted ore since it originated from the upper extraction tunnel which is geologically graded and controlled. However in the physical model, this rock was treated as 100% waste material. After 200 scoops were removed from the model (image c), the central ore band seen in image (b) is approaching the drawpoint opening and flushed into the heading. The higher recovery for this test can be directly attributed to the movement along the central axis of the ore column and retrieval of the upper apex. At 240 scoops, the end of the test run, almost all of the ore column was recovered. Over 95% of the dilution from this test run was from the cap rock, as evidenced in the drawpoint area of image (d). The remaining dilution is from the wall rock indicating that the particles from the back section of the ring were beginning to move into the drawpoint.
Figure 22 - Physical Model Test #17 (5.49 m burden at 80°)

Figure 23 shows model test 29 which yielded the lowest recovery value of all the 31 m sublevel interval trials. Image (a) shows the model near its initial state with 30 scoops of dilution-free ore removed from the model. After 90 scoops (image b) the ore remained dilution free. However, the waste from the upper drawpoint regions had not descended very far. Image (c) denotes the 30% instantaneous dilution cut-off point where the test run was to be halted. The image shows a fair amount of dilution in the drawpoint and all of this waste was from the wall rock. The cut-off point was obtained without dropping the upper ore apex and without recovering the long central zone of the ore column. Image (d) shows the model well beyond the standard cut-off point and into the 50% instantaneous dilution range. This photo further points out the high dilution present without the development of a strong central flow column. The
upper ore apex did not significantly drop between images (c) and (d) while the same held true for the waste rock contained within the upper drawpoints.

![Images showing physical model test results]

Figure 23 - Physical Model Test #29 (4.12 m burden at 85°)

The comparison between these two tests highlighted two main points in the design of sublevel cave operations with increased sublevel intervals.

1. Higher recoveries were dependent upon a strong central flow zone which recovered most of the ore within the long axis of the ore column. The flow strength of this central mass flow zone must be strong enough so that the upper ore apex descends to the drawpoint at the same time as the wall dilution. Recovering this upper apex is key to high recoveries.
2. The flow strength of the central zone was dependent upon the gradient of the ring. The closer that the ring gradient was to the vertical, the stronger was the vertical flow in the central zone. However, as shown in Figure 22, the combination of steep ring gradients and thin burdens produce low recovery values.

In summary, the near vertical ring gradients required for the high recoveries must be accompanied by the larger ring burdens. Thin burdens did not allow for full recovery of the ore column along its long axis since the waste from the wall rock, at the back of the ring, increased dilution to its cut-off point. Strong vertical flow was of paramount importance for increased sublevel heights and ore flow through the stope in general.
3.7 Description of Particle Motion Characteristic (PMC)

The Particle Motion Characteristic (PMC), is the form of the rock flow through the stope. The traditional notion behind the flow of fragmented rock through stoping areas has been to describe the form as an ellipse. The elliptical flow pattern predicts many of the traits of the PMC in a sublevel cave stope but has not been very helpful in optimizing its geometrical configurations. The reason for this is that the bulge or roundness of the flow pattern, particularly at the bottom of the ellipse, could not be measured accurately. The ellipsoid angle is flatter at the drawpoint opening where the draw cone is in a funnel shape and then it steepens to a near vertical angle. It is also known that the eccentricity of the ellipse increases with the height of the flow ellipsoid and therefore becomes less rounded. The confidence in estimating the degree of this rounding has never been high enough to transfer this knowledge to obtain a set burden size for a given layout. The rounding of the ellipsoid was dependent upon too many variables which changed for each individual blast and which was largely dependent upon particle size and ellipsoid draw height. The three main drawbacks for applying ellipsoid theory in the actual mine setting are:

1. The theory has not been clearly described to operation personnel in such a way that it is easily accepted. The reasons for the ellipsoidal flow form have not been totally understood in the actual mine situation as it has been developed as a result of physical model type experiments and this knowledge transferred into the mine setting.

2. The theory has not been specifically useful in the design of sublevel cave operations. It has provided excellent guidelines for mine design but in practice, lacks consistency. This deficiency is due to the sensitivity of the ellipsoid to variations in particle size distribution which may change radically between production blasts.

3. The theory can be described as a perfect mathematical shape. The ellipsoid's form, from an academic perspective, is simply a result of the geometrical configuration of the stoping area. This shows a strong dichotomy within the theory; on one hand the shape of the ellipse is clearly defined, on the other hand, it is dependent upon other more intangible factors such as particle size and uniformity. The fact that the ellipsoid theory does not incorporate all of the factors within the sublevel cave layout tends to weaken its validity as an effective design tool.
The study of the behaviour of the physical model and the data received from the test runs has led to the development of a different hypothesis for the behaviour of the PMC. Due to the efforts taken to represent true mine conditions with the use of the INCO physical model, it is believed that a more practical understanding of the gravity flow of solids has been obtained which can help in the design of sublevel cave layouts.

The concepts developed from this thesis, will be explained and supported by the results from the physical model. The basic concept for the Particle Motion Characteristic (PMC) theory can be summarized as follows:

1. The blasted rock from the stope follows a curvilinear plane, which will be referred to as the Sliding Plane (SP), to the extraction point. The form of the Sliding Plane may best be thought of as a conical cup with the apex pointing down into the drawpoint. The walls of the cup are straight and any bulging outside of this plane is considered to be inconsequential.
2. The two main controlling factors for the form of the plane are: the internal cave angle and the digging depth. The internal cave angle will be referred to as the Sliding Plane Angle (SPA). The SPA may be thought of as the angle of repose of the rock under a specific stress regime. The digging depth is defined as the effective depth of material removal into the body of the fragmented ore column. The degree of particle fragmentation within the ore column is also significant to the form of the Sliding Plane and is incorporated into the value for the SPA.
3. The Sliding Plane Angle is equal to the inclination of the dividing plane between movable and motionless particles. The digging depth has a direct bearing upon the horizontal placement of the SP. The sublevel cave design can then be improved upon based upon reliable and practical assessment of these two factors.

The SPA is controlled by the frictional characteristics of the caving material. The values of horizontal and vertical stress in that area are also important to determine the SPA. Of particular importance is the ratio of horizontal to vertical stress which changes along the length of the ore column. This stress differential was noticeable in the physical model tests and is the main reason for the surges of dilution. It is unlikely that this stress differential will be of any consequence in the actual mine setting. The horizontal stress acts like a confining stress in the ore column while the vertical stress is the flow stress. Improving the design efficiency will
depend on achieving the proper balance between these two stresses so that both waste fronts, from the wall and the cap, reach the drawpoint at approximately the same time.

The SPA attempts to incorporate the effects of particle fragmentation upon flow conditions. Consider that the particle size varies along the length of the ore column; finer sized particles near the collar of the holes progressing to larger sized boulders toward the toes. This fact is due largely to imperfect detonation along the entire length of the explosive charge combined with the lower powder factor at the toe of the hole. The finely blasted particles tend to produce more eccentric flow patterns while larger boulders produce flow ellipsoids that have a wider bulge and are more rounded. Particle size varies not only along the length of the ore column but also between blast rings. Even small operational differences between blast rings can change the particle size distribution. While this is true particularly for drilling, loading and blasting procedures, mucking rate is also a factor. Factors such as lag time between such operations and changing geological conditions will also produce subtle variations in particle sizes.

The ability of the particles to interlock or otherwise form a bridge during pulling should be incorporated into the SPA value. The physical models run during the course of this study indicated that interlock increased with a higher variability of rock particle sizes. This means that it is not only the large-sized boulders that present the problem of bridging but that an appropriate combination of particles will also constrict flow. Consider the situation wherein a group of mid-sized rocks are flowing down the stope. When a smaller particle filters into the only open space between these rocks, their mobility is seriously diminished, thus increasing the chance of interlock.

Considering all the variable factors involved in defining the PMC, it is clearly difficult to incorporate all of these intangible factors into one final number. The value for the SPA involves developing a holistic understanding and awareness of the many factors involved during the caving process and it requires that an average value be determined. The best way to develop this understanding is by continual observation conducted either by geologists or operation personnel. The foregoing statement acknowledges the practical aspect of the SPA and that instantaneous readings of this angle only give part of the picture. Instant readings are subject to the many variables which affect flow form; they present the SPA at a specific time, location and stress regime.
Developing an understanding for the behaviour of the flowing rocks in an actual stope is more a reflection of experience in the mine setting than a transference of results from a physical model. However, the results from the physical model indicate definite trends and similarities to the operation of the sublevel cave stope. These comparisons are useful in suggesting improvements to the stope design and do not incur high cost.

3.7.1 Analysis of the Particle Motion Characteristic

Upon blasting in the sublevel cave stope and during caving in the physical model, the contained particles undergo internal rearrangement. At that time, ellipsoids of loosening particles gradually develop and grow and increase their area of influence. The loosened area incorporates material in a defined region which has a lower bulk density. The shape of this region is typically ellipsoidal because the failure profile encompasses areas where originally, tightly packed particles are loosened and rearranged before being subjected to the direct force of gravity.

The particles within the stope resist failure primarily by the forces of static friction along their contacts. Another factor in this is the ability of the fragments to slide or roll to a less packed condition. For this reason the rock-to-rock surface friction is often of more consequence than the shearing resistance of particles since it is only the surface frictional resistance that will need to be overcome in order to affect flow conditions.

The highest static friction coefficients are developed when the primary stress acts perpendicularly to the main contact points. Particles in a static condition are considered to be in their initial stage of dilatancy where partial dilation of the particles occurs and is dependent upon stress distribution within the ore column. At this stage, contour envelopes of low density material develop and force redistribution sends very high lateral pressures to the abutments of the stope. The force redistribution will also tend to compact particles outside of the loosened envelope as stress is shed immediately to adjacent areas.

As mucking continues, the particles directly above the drawpoint opening collapse due to internal sliding between fragments. The process of freeing the fragments above the drawpoint occurs through translation. Collapsing particles near the drawpoint opening permit sliding of particles higher up in the ring. The region of sliding failure grows vertically and consequently develops low density dilating envelopes which pulse upward (Figure 24). When sufficient
dilation has occurred, adjacent particles are then loosened enough to be able to rotate and further dilate. The loosened particles then tend to flow towards the central sliding zone which then forms a conical failure profile. The stable angle formed by the cone failure is referred to as the Sliding Plane Angle (SPA). The development of the failure of the draw cones induces internal shear and rotational forces as the particles progress toward the drawpoint opening. The particles' frictional resistance to sliding and rotation then become a factor as the caving fragments are subject to both forces simultaneously.

Figure 24 – Mucking Process with Depression Zone Progression

A characteristic of the 'cave-in' process is that caving fragments are subject to variable sliding and rotational forces dependent on their location. The sliding motion is most apparent in the central draw channel (or zone of mass flow) while rotational forces are typically developed in the loosening envelopes which increase in size as mucking progresses. The various factors such as particle contact points and fragment size, involved in particle loosening, make it a very anisotropic process. The net effect of the varying frictional forces or network of forces produce a spider web-like aggregate with lowest resistance to failure parallel to the contact points.
(Figure 25). The aggregate body then moves both vertically and radially towards the central draw column. The combined effects of frictional resistance and the sliding and dilating forces form the loosening envelopes.

![Figure 25 - Aggregate Body of Particles with Network of Forces acting on the Contact Points](image)

**3.7.1.1 Formation of the Draw Channel**

The development of the motion ellipsoid is characterized by a series of expanding ellipsoids during the initial stages of material removal. The motion ellipsoid continues to grow in this fashion until a steady state flow channel is developed. The time it takes for the constant flow channel to develop is a function of the drawpoint width and the boundary draw conditions within the stope. The rate of descent of the particles within the motion ellipsoid is directly related to the change in bulk density of the yielding refill (Yenge, 1981) and the central flow zone is the region of greatest loosening and lowest density.
Particle freedom increases:

- toward the center line of the central flow channel and.
- downward to the drawpoint opening.

The particles between the flow channel and the limit ellipsoid reduce the flow rate of the central core as particles rotate and fall or otherwise obstruct the flow of the central draw channel.

The conical draw channel or draw cone is a direct representation of the cave-in process (Figure 26). The emergence of the draw cones causes material directly above the discharge outlet to be significantly loosened. Faulkner and Phillips (1935) describe this upward progression of loosening as a series of "Voussoir arches" which are successively loosened and eventually collapse. As the arches collapse, a chimney channel or depression zone moves upward. This region is the central part of the draw cone. The stress from arches, which were once stable, is gradually transferred to the abutments of the stope. In accordance with this rationale, *hang-ups* are the result of uncollapsible Voussoir arches.
As loosening within the limit ellipsoid progresses, the refill material enters the corresponding yielding zone. The region between the draw channel and the limit ellipsoid at the lower end of the ellipsoid is mainly influenced by radial expansion and rotational forces (Figure 26 – upper diagram). This will be referred to as the Obtrusion Zone. There is a large
difference between the rate of change of bulk density between the Obtrusion Zone and the central draw channel. This difference accounts for the dissimilar respective discharge rates. A narrower Obtrusion Zone will result in a flatter domed rupture profile or a profile of failing Voussoir arches which yield toward the ends of the beam rather than the centre (Figure 27).

Figure 27 – Draw Experiments with Compact Sand and variable Opening Sizes (Stazhevsky, 1990)

The overall loss of strength and the effects of loosening are particularly noticeable at the discharge opening, hence the higher flow rate. The action is accompanied by a series of collapsing Voussoir arches which exert a reconfining stress upon the lower particles. Therefore, it can be concluded that it is actually particle dilation which is the controlling factor for the internal flow behaviour of rock. This has been proven in the mine setting. The region of maximum particle dilation occurs within the converging section of the draw channel. This fact, combined with the funnelling effect, make this throat region most susceptible to particle hang-up.
Within the flow pattern there are also regions of compression. The end result of these conflicting forces produces sudden and dynamic flow changes to the aggregate falling body of particles.

The draw cone narrows to approximate the size of the discharge outlet near the bottom of the stope for the following reasons:

- The direct exposure to gravitational force exerted on the particles is equal in width to the drawpoint opening. The particles' resistance to the direct force is low.
- The relative rate of change in bulk density between the central core and the feeder zones becomes more pronounced near the drawpoint opening, creating a definite boundary between moving and stationary particles.
- The strong vertical motion of the central core places a confining stress upon the peripheral fragments outside of the draw cone, increasing their resistance to failure. The confining stress increases with overburden height, hence its value is higher toward the bottom of the stope area. The confinement restricts dilation which is necessary for loosening and movement.

The draw channel then widens upward from the drawpoint as the influence of the drawpoint opening causes a wider range of the particles within the limit ellipsoid to become loosened. The width of the central core has a direct impact upon the horizontal width of the particles which were affected by loosening. The effects of gravitational force, rate of change of bulk density and confining stress on the peripheral areas are lessened upward from the discharge opening. This allows the draw cone to widen and become stable when the width of the limit ellipsoid is reached because the relative strengths of sliding (direct) and rotation (indirect) motion become equal with respect to their influence upon the horizon of particles.

The depression zones do not move solely along a plane of potential failure. The rupture zone (or region of break in the Voussoir arch) is dependent upon:

- The inherent behaviour of the caving material (friction values, packing, uniformity, etc)
- The boundary conditions and size of the discharge opening and,
- The effects of particle expansion upon the material.
3.7.1.2 Internal Cave Angle

The internal cave angles described by Yenge (1981) represent the angle of the boundary between stationary and mobile particles within a specific stress regime. In this sense, the internal cave angle is similar to the Sliding Plane Angle. Observations from Yenge's tests, conducted on a uniform size distribution of particles, depict that the internal cave angle steepens upward and outward from the drawpoint opening (Figure 28) until a vertical angle is achieved. This effect was also noted by Kvapil (1965) and earlier by O'Calaghan (1960) who stated that "the failure line between moving and stationary grains was curved and broke (upward to) the free surface in a vertical direction."

![Diagram of Internal Cave Angle](image)

Figure 28 – Variation of Internal Cave Angle with respect to the Boundary Plane of the Draw Channel (Yenge, 1981)

The rounding, or change in angle near the bottom of the draw channel was evident in the model tests conducted at INCO. It was observed that the internal cave angle is largely dependent upon the internal friction angle of the material. The internal cave angle is dynamic in nature, changing according to particle size distribution and stress regime. These traits account for its
variability along the vertical axis of the draw cone and at different stages of unloading. The internal cave angle should be characterized as the *instantaneous* boundary between stationary and moving particles whereas the Sliding Plane Angle is the resultant boundary for an individual test.

Although, the physical model demonstrated the development of the internal cave angle, the Sliding Plane Angle became the obvious and dominant factor when stable draw conditions were reached. The draw conditions became stable after the central draw column became established, after the removal of the first one-third of the total buckets.

### 3.7.2 Defining the Sliding Plane (SP)

The Sliding Plane Angle (SPA) is a useful design tool because its value serves to:

- evaluate the myriad of factors involved in determining particle flow in the sublevel cave stope. The effects of the intrinsic nature of the particles as well as the stress regime and stage of pull are simplified through the SPA.
- present a holistic and stable evaluation of the boundary between moving and static particles.
- incorporate practical knowledge into the assessment of the particle motion characteristic, accounting for regional or operational concerns specific to the stope site.

The concept of the Sliding Plane (SP) is agreeable to the actual mine conditions and to the results obtained from the physical model. The SP is the result of the combined influence of the central draw cone (gravitational force) and the Obtrusion zone (rotational and dilating forces). Rather than separating into two distinct regions, the SP accounts for the possible movement similarities of the two zones. For example this considers that the portion of the Obtrusion zone closest to the draw channel is also subjected to a relatively high component of gravitational force when compared to the more peripheral portions of this region. Similarly, the effects of gravity is definitely the motion which controls the movement in the centre of the draw column. In the physical model, the result of the central channel was the establishment of a stable SPA.
Formation and grade of the SPA is determined by:

- Discharge outlet dimensions
- Particle size distribution and uniformity
- Particle surface friction angle
- Particle intrinsic material properties (density, strength, moisture content, etc)
- Superincumbent stress or Overburden load

3.7.2.1 Effects of Particle Size upon the Sliding Plane

Consider the formation and grade determining factors above, the only way to affect the SPA is by controlling ore column fragmentation and the width of the drawpoint opening. Finer particles increase the SPA due to their more eccentric flow pattern, due to their low void ratio and high unit weight. These factors enable movement early in the drawing process because of the relatively small expansion required for motion. As a result, a large proportion of the smaller particles within the central draw column are immediately affected by gravity forces (Figure 29). The large differential in gravitational force between stationary and falling particles creates a definite boundary and greater the height of the loosened particles creates a steeper SPA. The effect of particle size and dilation upon flow pattern becomes obvious when one considers that a larger fragment with 2 times the diameter of a smaller piece requires 8 times the void space to allow for expansion. This is due to the cubic relationship between particle diameter and the resulting volume differential.

Larger sized fragments with high void ratios and relatively low unit weights produce more rounded flow patterns. The larger particles need to expand a greater amount, when compared to their smaller counterparts, resulting in a higher rate of change in bulk density for the ore column. In the central draw channel, those large fragments immediately above the discharge opening are able to expand and move, gradually decreasing the amount of available void space there. The decrease in void space inhibits the dilation of the large particles and creates a hypothetical horizontal boundary, within the central flow column between confined and dilating particles. This will be referred to as the Dilation Boundary (DB). The DB is also evident with smaller sized particles but would be found at a higher vertical level than that of the large particles.
The flow pattern and form of the SP become less consistent and more difficult to predict when considering a heterogeneous assortment of flowing fragments. A non-uniform mixture of particle sizes would tend to increase the degree of particle interlock, thus:

- decreasing the flow rate of particles
- increasing the SPA
- decreasing the width of influence of indirect forces, which decreases the range width of refill particles (which decreases possible stope widths)
- lowering the vertical elevation of the Dilation Boundary
3.7.2.2 The Dilation Boundary (DB)

The effects of direct motion (gravity) decrease as the vertical distance between the drawpoint opening and the particles’ location increases. Concurrently, the influence of indirect motion (rotation) increases. Particle movement and dilation have a constitutive relationship in that one cannot occur without the other and vice versa. Dilation occurs as a result of moving or removing particles; and particles only become movable when they have dilated sufficiently to be subject to motion forces strong enough to cause failure within the assemblage.

Radially outward from the centre line of the stope decreases the effective magnitude of direct force. Transferring the motion/dilation relationship further from the centre makes it apparent that the Sliding Plane Boundary has similar characteristics to the Dilation Boundary. The difference between the formation of the two boundaries is that the field of influence for the Sliding Plane (SP) is radially out from the draw column’s centre line while that of the Dilation Boundary (DB) is vertical.

The Dilation Boundary is more definite than the Sliding Plane since it is perpendicular to the direct force. The formation of Voussoir arches between stationary and mobile particles also contributes to the distinctiveness of the DB. The SP is less distinctive because indirect forces are less consistent during the pulling procedure and the effects of gravity are not equally applied along the entire length of the boundary. Direct forces are highest upon the section of the plane which is closest to the drawpoint opening. The force direction at this point is roughly parallel to the Sliding Plane and the particles are actually sheared right off the boundary. This accounts for the free fall action of fragments in this region. Moving radially out from this chimney channel reduces the strength of the direct force upon the SP boundary. Further up the plane, the boundary is affected by the translation of the direct force through the fragmented rock mass. In this sense, the SP is affected only by the ability of the direct force to induce indirect motion upon the particles located linearly between it, and the chimney channel.

The SP begins angling away from the discharge opening as the immediate effects of gravity diminish. The particles between the vertically flowing chimney channel and the SP boundary are subject to the combined effects of direct and indirect forces. The magnitude of the direct force upon the SP decreases as the boundary moves up and away from the discharge opening. The loss in direct force magnitude creates a less definite boundary as particles are less subject to the shearing action of the direct force and more controlled by the rolling action of the
indirect force. The SP represents the point at which the force combination does not allow for movement.

Due to the increasing length of the chimney channel, the width of particles affected by indirect forces increases linearly in an upward direction away from the discharge opening. Increasing the length of the source of direct motion increases the number of particles so affected. This is because the force from the chimney channel results in a widening of the band of rotational motion. This force transference influences the upper particles in both a radial and vertical direction. The addition of these two forces would result in a vector direction parallel to the SP with increasing strength as it approaches the chimney channel. The compounding effects of the chimney channel widen the zone of motion. This is because a greater number of particles are subjected to a combination of force factors which enable movement. The widest section of the funnel is characterized by a horizontal band of particles influenced by rotational forces which closely overcomes the surface contact friction.

3.7.2.3 The Interlock Zone (IZ)

The near stagnant condition of the widest region of the funnel increases the degree of interlock for particles which are located above this area. The slowing of particle motion and the reduction in direct force magnitude decrease so that particles are unable to flow. In this section, it is the rotational forces which persist that cause the particles to roll and slip into critical void spaces thereby increasing the packing factor and inhibiting dilation. The combination of these factors in this near static region make it likely that Voussoir arches will be formed and prevent particle flow entirely.

The effect of confinement is cumulative so that the band of motion decreases linearly past the funnel’s widest section. The profile of mobile particles at the top section would be shaped like an A-frame rooftop. The re-confinement of particles and formation of Voussoir arches at the funnel’s widest section prevent the indirect force created by the lower chimney channel from influencing motion. The rotational forces instilled by the lower chimney channel do not end abruptly, but the influence of indirect force is reduced beyond the funnel’s widest section.

The direct forces associated with the central chimney channel persist all the way up the ore column so that particles in this channel are still subject to gravity force. The flow of the particles in the upper chimney section does not occur immediately since it is dependent upon the
dilation and movement of particles below. The establishment of this central channel in the upper region of the column introduces direct and indirect forces to the region beyond the funnel’s broadest section. The magnitude of these forces is not as strong as those of the lower section due to the channel’s reduced area of influence and dilation. The combined effects of the loss in rotational forces, accompanied by the reintroduction of direct forces, explains the A-frame rooftop shape of mobile particles in this upper region (Figure 30).

Figure 30 – Interlock Zone
The reduction in motion inducing forces at the funnel’s broad or interlocked section decreases the band width of mobile or ‘feeder’ particles. The feeder particles are those moving rocks which migrate to the central draw column. This decrease was not evident in most of the physical model tests since interlock of the fine uniform particles rarely occurred. The two cases in which a small degree of interlock did occur resulted in increased recoveries. In these cases, the interlock restrained the flow of the feeder particles which was the waste material from the upper drawpoints. This hindrance of the waste material allowed for the central chimney channel to develop to the upper region of the column and recover the ore from the upper apex area.

The degree of interlock in this same region of the actual mine stope is expectedly higher than that of the physical models. The high heterogeneity and non-uniformity of particles accompanied by the discontinuous and abrupt particle dilation in the mining cycle greatly increases the likelihood of particle bridging and interlock. This would constrain the width and flowability of the feeder zone and would assure the development of the chimney channel to the upper stope region. The feeder or refill region is just beyond the interlock zone.

The lower Interlock Zone reduces the width of the feeder zone. This reduced width would require that the sublevel cave design not have a large stope width. The writer would propose that developing a proportional relationship between drawpoint width or sublevel interval and stope width is unwarranted in the actual mine setting for the following reasons:

- A zone of high particle interlock is almost guaranteed in the sublevel cave stope. The variability in fragment size from the bottom to the top of the blasted ore column ensures particle heterogeneity. The larger fragments found towards the toes of the holes (or surrounding the misfired holes) will become mobile as the chimney channel develops. This will decrease the uniformity of the caving rock as these large pieces mix with those found in the lower sections.

- The discontinuous action of the mucking cycle gives the particles time to re-pack to a more stable and static condition. Also, in headings where the drawpoint width is wider than the scoop bucket, the void space created is not consistent during the entire pull. This decreases the uniformity of the direct force's direction.
The ellipses of motion within the caving stope are not created instantaneously, they develop as a result of the gradual withdrawal of material. Accordingly, the particles in the upper region may be suspended for long periods of time until they are subjected to motion forces. This problem becomes even more evident when considering the production schedule which may leave stopes suspended for months. The effects of oxidation to sulfide ores due to the increased time lag would also promote arching in the stope.

The Interlock Zone overrides the influences of the increased sublevel height and the width of the drawpoint opening. This has not been witnessed in the physical model but the writer believes that the conditions in the mine environment will create strongly bridged zones. The physical model, with its near non-existent particle interlock, proves that there is a linear relationship between drawpoint and stope widths. Therefore it is not surprising that sublevel cave designs based upon physical model results advocate wider stopes for both increased sublevel heights and drawpoint widths. The effects of an Interlock Zone or similar region of arch formation were not evidenced and, as such, were given little consideration.

The writer suggests that the Interlock Zone would be located at the widest section of the draw funnel due to the relatively high indirect forces which do not allow for motion. It is believed that this zone must be at a height of at least 50% of the sublevel interval in order to attain adequate recoveries. The location would be highly dependent upon particle size, the frictional properties of the material, particle uniformity, overall stress regime and flow continuity in the ore column. It is conceivable that a change to any one of these factors that would seriously increase interlock could actually necessitate a reduction in stope width.

The vertical elevation of the Interlock Zone is another variable that is highly dependent upon factors which tend to be inconsistent in the development of the sublevel cave stope. These inconsistent factors include blasting, discontinuous mucking, stress conditions and geology. The Interlock Zone has a bearing upon optimizing the stope width but its location and degree of influence may only be defined through practical assessment and observation.
3.7.2.4 Relationship between Drawpoint Width and the Sliding Plane

The primary influence of the extraction heading upon ore flow is that it provides the direct outlet during the course of pulling from the drawpoint. This effectively defines the area of the direct forces which set the ore column in motion. Wide, flat back headings are the most desirable for higher recoveries since a greater percentage of the ring's cross-sectional area will be situated directly above the opening. The dimensions (width × burden) and shape of the drawpoint determine the bounds for the chimney channel. In actuality, the size of the chimney channel, which is the region of mass movement within the central core, is determined by the heading width and the effective digging depth of the mucking machinery.

The chimney channel, or zone of mass flow, defines a flow pattern which yields primarily due to the shearing action induced by the direct forces. Additional characteristics of the mass flow zone include:

- The width of its central channel expands at the expense of the Obstruction Zone (radial feeder zones) thereby increasing the rate of descent of materials in the flow ellipsoid.
- The downward flow of material within the central core is opposed by the radial shearing resistance of the adjoining Obstruction Zone. The resistance interaction causes forces to be transmitted all the way to the limits of the draw cone where secondary (rotational) forces control motion.
- The movement of the mass flow zone is essentially vertical. The direct force magnitude within the central column at a given level is essentially equal. Therefore, the loss of strength in the yielding material at that particular elevation will be virtually the same. It then follows that the rate of change in bulk density within the mass flow zone is less than that of all the other non-core flow regimes.
- The superincumbent load upon the central core affects the SPA and the size of the radial feeding Obstruction Zone. At the onset of flow conditions, the entire weight of the superincumbent load is placed upon the discharge opening. Increasing this weight increases the flow strength of the mass flow zone creating a higher distinction between particles set in motion by direct forces, compared with those controlled by indirect force.
- Similarly to the superincumbent load, the drawpoint width increases the area of effect, hence, the direct force magnitude within the central core. The high magnitude of direct force within the mass flow zone creates a region of definite shearing failure. With the higher direct force
magnitudes, the failure occurs suddenly and does not allow for much radial shear resistance to develop. This in turn decreases the development of secondary forces within the draw cone.

The two main factors to mass flow conditions are the size distribution of particles and the proportional magnitude of the direct forces within the central core. The direct force magnitude of the chimney channel is a function of the superincumbent load and the effective size of the drawpoint opening created during mucking. The drawpoint dimensions are the only design factors which may control the strength of the direct forces in the central core in practical terms; so the effects of the superincumbent load will not be discussed at length. It is sufficient to say that sublevel cave layouts at great depth may require changes from those at shallower depths.

The drawpoint opening is a controlling factor in determining the boundary conditions of the mass draw movement of the particles within the stope. This mass draw may also be achieved by increasing the ratio of the drawpoint area to particle size. This further emphasizes the importance of blast results to sublevel cave operations. A finely blasted ore column increases the flowability not only of the mass flow zone but also the entire stope as a whole.

Once mass flow conditions are met, further increases to discharge opening dimensions or superincumbent load will not lead to an increase in flowability. This fact places a limit upon the necessary size of the outlet opening. Moreover, flow problems cannot always be controlled by simply increasing drawpoint widths, burden sizes or digging depths.

Increasing the size of the core mass flow zone allows for quicker dilation and movement of the particles in this region. Secondary forces do not have time to develop within the broken rock mass within the stope area. As a result, the SPA increases with increasing discharge widths or openings. The immediate application of direct force controls the yield behaviour of the rock fragments in the central channel and makes a more distinct cut between falling and stable particles.

The wider drawpoints effectively increase the loading rate upon the rock fragments in the central channel. The increased rate creates a very high initial direct force value which causes a shearing action to the rock in the central channel. The magnitude of the force is high enough to cause a definite break between particles and occurs suddenly so that the secondary forces are unable to overcome the inertia of the surrounding rock fragments. This break reduces the size of the Obtrusion Zone and increases the SPA.
The limit to which increased drawpoint widths will increase the SPA is believed to be very close to 90 degrees in the actual mine environment. The wider drawpoints increase the strength of vertical flow and the size of the central core channel. This fact is very beneficial if you consider, as the writer does, that this central channel is the only region which may be recovered with a high degree of confidence. Recovery and flow strength of the central core become even more important as sublevel intervals are increased.

A comparison between the widths of the mass flow zones is shown in Figures 31 and 32. The only difference between these two tests is the width of the discharge opening. The 5.49 m opening shows a strong vertical flow which is evident by the steep and definite boundary between the black and white rock. The 4.42 m drawpoint shows that secondary forces are a more important factor because of the roundness of the descending black rock.

Figure 31 – Mass Flow Zone for 4.42 m Wide Drawpoints
Defining the boundary between the two rock types is not as clear as it is in Figure 32 because the direct force does not predominate in causing the yield zone as much as in the 5.49 m drawpoint. The direct force differential between the two drawpoints dictates the form of the caving rock to the opening. Results from the physical model indicate that the SPA increases with drawpoint width. The 4.42 m drawpoints had an average SP angle of 75°, while the 5.49 m drawpoints had an average angle of 80 degrees. These angles are determined from only the 4.12 m and 5.49 m burdens because these were the only sizes tested with the 5.49 m wide drawpoints. The 1.07 m increase in drawpoint width results in a 5° increase to the SP angle and it is believed that this value is quite close to the maximum angle possible in the physical model. This angle would approach 90° in the mine setting due to the increased degree of interlock that could be expected.
The angle of the SP is particularly important when determining burden sizes. The Sliding Plane originates as a result of the motion from the central flow channel. This central channel is directly within the burden, making the SPA critical to optimizing burden geometry and inclination. Conceptually, the ring gradient would be equal to the SP angle. The difficulty with this plan, however, is determining the location of rock withdrawal or mucking point. The effective digging depth of the scoop tram controls the position of the mass flow zone which, in reality, would be toward the front of the burden. The effective digging depth is the distance past the brow that the scoop bucket is able to pass into.

3.7.3 Substantiating the Sliding Plane Hypothesis

The concept of a Sliding Plane is supported by the findings from the physical model in two ways:
1) The linear and distinct boundary between mobile and stationary particles is clearly visible from the front view of the model. These flow lines are consistent in their positioning and inclination for matching test configurations and material.
2) The results from the tests indicate a definite trend with respect to dilution location and burden size and inclination. This data points to a definite plane of slide within the ore column which can be traced from the manner in which waste dilutes the caving rock stream.

The existence of a Sliding Plane results from the internal forces within the flowing stream of rock material. Therefore, the SPA is different from the angle of repose of the same material. A stationary heap of material is governed only by the static friction between the contacting fragments while the SPA considers the dynamic nature of the draw conditions and flow. Richards and Jenike clarified this situation when they wrote:

The essential framework point about the angle of repose is that it is relevant only to a heap at rest. Attempts to relate this angle to the dynamic situations are a mistake. (Richards - 1970)

The angle of repose assumes values between 30° to 40° and it is nor a measure of the flowability of solids. In fact it is only useful in the determination of the contour of a pile. (Jenike - 1964)
Also, the angle of the SP should not be related solely to the internal friction angle of the material. This would presume that the material in the draw column behaves much like a solid body. This is clearly not the case since the caving material does not move along weakness planes as they do in a fissured rock mass. Instead, it is the boundary conditions dictated by the drawpoint opening and the intrinsic nature of the flowing rock which control the yielding pattern, and hence the Sliding Plane.

Validation of the SP in the physical models is accomplished by observing the flow behaviour and accounting for the source of the dilution for the individual test runs. The findings from these trials are best represented in Figures 33, 34 and 35. These diagrams illustrate the tested ring burden sizes with ring gradients set at 75, 80 and 85 degrees with a sublevel interval of 31 metres. The line through the columns indicates the expected position and inclination of the Sliding Plane for the 4.42 m drawpoint widths. The aspect represented in the figures shows a side view through the mid-section of the burden. From this perspective, the location of the plane would be at its greatest horizontal distance in the ore column.

In all cases, the effective digging depth is held constant at 1.83 m and the SP angle is considered to be 75 degrees. The average SPA for the physical model tests with sublevel intervals of 31 m and 4.42 m drawpoint widths is concluded to be 75 degrees. Best efforts during the simulated mucking of the model attempted to keep the digging depth to 1.83 m (or 3.8 cm actual) into the drawpoint. Maintaining the SP angle and the digging depth constant at the designated amounts for the various burden sizes provided results that were consistent to the concept of a Sliding Plane.

Figure 33 shows two 2.74 m burdens inclined at 75 and 80 degrees. The boxes positioned to the top left of the diagram indicate the recovery rank, recovery amount and the dilution source for the indicated burden when the cut-off point of 30% instantaneous dilution has been reached. The recovery rank is considered more important to determining improved geometrical designs for the purpose of these tests. The value of recovery provides a direct quantitative measurement but it is difficult to gauge the significance of its value. For instance, the difference in recovery between the 2 configurations shown in Figure 33 is only 4 percentage points (or 7%). The burden inclined at 75° provides marginally better results with respect to the amount recovered. The recovery figure may be misleading in this sense since the value of improving the recovery by 4 percentage points in the physical model is not clearly understood. The conclusion drawn from these findings is that the 75° ring provides better results.
Figure 33 – Sliding Plane in the 2.74 m Burden

Recovery = 64%
Dil. Source = 95% Wall

Rec. Rank = 1
Recovery = 60%
Dil. Source = 100% Wall

Burden Width

Sublevel Interval

Ore/Waste Boundary

Sliding Plane

Ring Gradient

Digging Depth

Drawpoint Height

Sliding Plane Angle
In collaboration with the recovery values, the location of the principal dilution source substantiates the presence of the SP. The location of the SP for the 75° ring in Figure 33 essentially runs along the ore/waste contact. In comparison to the 80° ring, it is evident that better results would be expected from the 75° ring due to the location of the SP. In both cases, most of the dilution is from the wall rock but the SP at 80° ring cuts into the waste area much sooner than its flatter counterpart. The size of this wedge of waste becomes increasingly noticeable further up the ore column. This wedge, resting on top of the SP, can be expected to enter into the descending draw channel. This explains the complete dilution from the wall rock and coincident lower recovery value.

Figure 34 shows the 4.12 m burdens inclined at 75, 80 and 85 degrees. It is observed that recoveries for these configurations decrease with increasing ring gradients, although this concept does not provide a useful explanation for understanding the design. For this, it is necessary to consider the inclination and placement of the SP.

For the 4.12 m burdens, the flatter rings provide better recoveries because the SP either does not enter into the wall waste region, or enters at a higher vertical level. Almost 75% of the 75° ring burden is enveloped by the Sliding Plane which then runs parallel to the ore/waste boundary. The initial point of entry of the SP into the ore column will be referred to as the Burden Intersection Point (BIP). The BIP determines the amount of the burden initially encompassed by the plane, its value a function of the digging depth and SP angle. The evaluation of the location of the BIP indicates that the waste for this set up should all be from the cap. This is not the case, since 20% of the waste results from the wall rock. This can be explained by the development of an ellipsoidal shape within the physical model. The ellipsoid flow form is also noted in the results of the 80° ring, wherein all but 5% of the incoming waste is from the adjacent wall rock area.

The similar recovery values found between the 75° and 80° rings can be explained by the incomplete inclusion of the ring by the SP for the flatter ring and the achievement of the SP to the wall waste area in the steeper ring. These facts balance each other so that comparable recovery results are achieved.
The relatively large difference between recovery values occurs with the ring gradient set to 85 degrees. For this configuration, recovery drops by 20% compared to the recovery achieved with the 80° ring. The emergence of the SP into the wall waste for the 85° ring occurs at a vertical distance of approximately 5.5 m before its flatter counterpart. This difference in vertical intersection into the ore/waste boundary is considered significant as the wall waste accounts for 100% of the dilution and the subsequent recovery value decreases.

Figure 34 – Sliding Plane in the 4.12 m Burden
The 5.49 m burdens are shown in Figure 35. The highest recovery from all of the 4.42 m drawpoint tests is achieved with the rings inclined at 80 degrees. For this configuration, the SP enters the wall waste region 3 m from the top of the burden. The long and thin wedge of ore between the SP and the ore/waste boundary would be partially recovered from the flow ellipsoid. The fact that 95% of the waste originates from the cap indicates that there is a buffer zone of ore between the SP and the ore/waste boundary. The entrainment of wall rock to the dilution probably occurs from the widest and most rounded section of the flow ellipsoid.

![Figure 35 - Sliding Plane in the 5.49 m Burden](image)
The ring inclined at 85° yielded similarly high results as compared to that angled at 80 degrees. The 50/50 split between waste sources seemed to optimize the flow pattern so that the movement from the top and from the side incorporated equal amounts of wall and cap waste. In this sense, the convergence of the two waste fronts at the same time would provide the highest recovery as ore from both regions is pushed to the drawpoint. The recovery value for the 85° ring gradient may be considered an efficient configuration, as it ranked second highest among all 4.42 m drawpoint tests conducted.

The near vertical inclination of the 85° rings promotes the magnitude of direct forces and vertical flow within the ore column. For the 5.49 m burden, the SP intercepts the ore/waste boundary at the mid-height of the 85° ring. Assuming even flow from both dilution sources, the cap rock would be 30.48 m away from the drawpoint while the wall rock is 5.49 m away. Given this evidence, the speculation with the 85° rings is that, the vertical force is approximately five and a half times greater than the horizontal force to the drawpoint. The 80° rings would have a lower vertical to horizontal force ratio due to its deviation from the vertical axis. This fact rationalizes the higher recovery found for the ring inclined at 80 degrees. The vertical level of the intersection point of the SP and the ore/waste boundary, in conjunction with the ratio of vertical to horizontal force, determine the flow of waste fronts to the drawpoint.

The magnitude of the vertical force is maximized when the ring gradient is 90° but the horizontal force never reduces to zero. Increasing the magnitude of the vertical force effectively:

- increases the angle of the Sliding Plane and,
- increases the eccentricity of the flow ellipsoid.

Optimizing ring recoveries is accomplished by combining the resultant vertical to horizontal force ratio (V/H ratio) to the appropriate intersection point of the SP with the ore/waste boundary or Plane Intersection Point (PIP). Both the V/H ratio and PIP are indirectly controlled by geometrical configuration of the layout.

The 5.49 m burdens, due to their wide band of highly uniform particles, are believed to further promote vertical flow. The smaller particles with their lower dilation limit improve the development of the chimney channel which enhances direct forces in the ore column. This fact was not readily apparent in the observations made from the physical model tests. The reduced
scale of the model may have an appreciable effect upon the increased vertical flow due to uniformity and reduced particle size.

The most important factor with regards to the V/H ratio was the vertical elevation of the PIP. Locating this point in the upper portion of the 80° rings is particularly important to achieving high recoveries. Comparing the dilution source between 80° rings with burdens of 4.12 m and 5.49 m, the PIP for the 4.12 m burden occurs at a point roughly equal to 40% of the ring’s height. The same point for the 5.49 m burden occurs at a vertical distance almost equal to 90% of its height. The difference in PIP vertical elevation explains the 95% cap rock evident for the 5.49 m burden compared to 5% cap rock in the 4.12 m burden. The lower recovery value for the smaller burden can be accounted for by the deficiency of ore recovery in the upper portion of the ore column.

The ring inclined at 75° yielded the lowest recovery results for the 5.49 m burden set. In these results the SP runs parallel to the ore/waste boundary but the entrance point of the plane (or BIP) is only 3 m into the burden, leaving about half of the burden uncovered by the SP. The recovery value for the 75° ring remains relatively high but the unrecovered ore substantially decreases extraction efficiency, making it a less effective configuration. All of the waste for this configuration is from the cap rock; attesting to the cushion of ore remaining between the SP and the ore/waste boundary. Improving the recovery of the 75° ring would be accomplished by increasing the effective digging depth. This would move the BIP deeper into the drawpoint so that a greater proportion of the burden would be encompassed by the plane.

Encompassing a high proportion of the burden in this fashion is necessary for ring gradients equal to the SP angle and highly recommended for steeper inclinations. The digging depth contributes to the size of the immediate opening created at the drawpoint. It dictates the depth of the chimney channel, which in turn is important to increasing the magnitude of the vertical force within the ore column. For these reasons, an effective sublevel cave design requires that the SP encompasses a high proportion of the burden at the brow level of the drawpoint area. Consequently, overly large burdens would not provide higher recoveries since ore remains unrecovered towards the back of the ring and vertical movement is not induced in a high percentage of the burden.
3.7.3.1 Marker Recovery

The coloured and labeled rock markers placed in the physical model provided a useful tool in determining flow progression within the ore column. The markers were placed at representative 4.6 m intervals beginning at a height of 6.1 m from the base of rail (B.R.) of the extraction tunnel. The markers divide the ore column as shown in Figure 36, which represents a stope with 5.49 m burdens and drawpoint spacings of 12.19 metres. This view illustrates that the markers are set to divide each individual stope block into four equal sections.

The middle marker is in a position which is central to the extraction heading. The markers to the side of the centre are spaced 3 m apart. The markers within the ore column are placed equi-distant to half of the burden while another row is set along the ore/waste contact. These strategic locations will help determine the 3-dimensional flow behaviour of the rock within the physical model.

![Figure 36 - Marker Location](image-url)
The recovery of the markers during the tests provided further proof as to the existence of a Sliding Plane. In general, the central markers were recovered sequentially in order of vertical placement. The lowest central markers (at B.R. + 6.1 m) located along the ore/waste contact, were usually not recovered until the end of the test pull. This indicates that the flow at the bottom of the ore column is relatively fast along the ore/solid rockwall contact. This observation supports the theory presented in this thesis which describes the lack of force development and translation near regions of strong direct force, as is the case at this immediate outlet area.

The markers along the sides of the stope at a vertical elevation of B.R. + 15.24 m, or 11 m within the ore column, were typically only recoverable from the tests. This fact complements the SP theory since the development of the slide to a distance of 3 m out from the centre line occurs only at a set vertical distance from the drawpoint opening. This provides proof that the SP has a constant angle at least to this point. Consider, SP angle = 90 - arctan (3 m/11 m) ≈ 75°; and this was found to be typical for SP angles.

The side markers between the vertical interval of B.R + 15.24 m and B.R. + 24.38 m were usually recovered. Beyond this limit, only the centre markers could be counted on to be recovered. This indicates that the feeder zone to the draw channel becomes narrower past the widest section of the draw cone. The effects of particle interlock accompanied by a decrease in direct force magnitude were shown by the loss of side markers above the B.R. + 24.38 m mark; this confirms the re-consolidation of the flowing particles past the widest section of the flow channel. This effect was previously described in Section 3.7.2.3; The Interlock Zone (IZ) of this report. However, it is the characteristics of the marker recovery which provide evidence of the existence of an interlock zone even within the physical model.

The markers used in the tests were useful in determining trends for the flow behaviour of rock within the physical model. It is interesting to note that no two tests, even with identical configurations, yielded the same sequence of marker recovery. This indicates that there is a degree of randomness and strong intangible factors involved with every test pull, even with the most tightly controlled laboratory conditions.
3.7.4 Surge Effect

The term 'surge effect' defines wide variations in the percentage of dilution entering the ore stream. The presence of a surge suggests that dilution to the caving ore column does not occur evenly. Even waste entrainment would show that dilution percentage increases linearly at a certain point during the mucking process. Surges present large inflows of waste at a certain point which are typically followed by a large inflow of ore.

The most significant surges in the physical tests were found with the 4.12 m burdens. The burdens and associated dilution to recovery graphs are shown in Figures 37, 38 and 39 for the varying ring gradients. The relationship between dilution and recovery for the 75° rings is relatively stable. The lines from all 3 drawpoints are quite smooth and linear as dilution increases gradually with recovery. The surge effect becomes more pronounced as the gradient of the rings increases. The lines on the dilution to recovery graphs for the 80 and 85 degree rings become increasingly undulated and rough. The pulsation of the waste entrainment in this fashion indicates that surging is taking place.

The only difference between these three figures is the ring inclination of the ore column. In the physical model, the increasing ring gradients would expose a larger percentage of the burden to direct vertical force, as indicated in the figures. This would create a stressed column of rock within the ore column and compact a larger proportion of the Sliding Plane. The stressed column has high vertical forces acting upon it since it is directly subjected to the force of gravity. This column would then have a greater tendency of being flushed to the drawpoint in relation to the surrounding particles.

The stressed ore column creates unstable draw conditions which become increasingly noticeable as the stress region moves down to the drawpoint opening. This occurs in the 85° ring where there is evidence of at least two distinct occurrences of surge. The stressed column has two major effects upon draw conditions, it:

- Promotes vertical flow within the central portion of the burden. This trait becomes particularly beneficial and perhaps necessary for situations of obstructed ore flow.
- Prioritizes the flow of rock from the central portion of the burden, as the force of gravity acts to push it towards the drawpoint opening.
Figure 37 – 4.12 m Burden at 75°

Figure 38 – 4.12 m Burden at 80°

Figure 39 – 4.12 m Burden at 85°
The flushing of the central portion of the ore column explains the high pre-dilution recoveries obtained with the steeper ring gradients. The low horizontal force eventually does move an initial load of waste into the region of the stressed column. This horizontal action is largely inconsistent since it is difficult for the waste to enter into the consolidated stressed ore column. The entrance of waste material into this column is likely due to the stop and start nature of flow imposed by the mucking cycle.

In the physical model, the direct exposure of the ore column to gravity force significantly affected results for surge and flow of dilution. The effects of the stressed column will not likely be as great a consequence in the actual mine environment. The high degree of particle heterogeneity and decreased number of contact points between ore particles will prevent the creation of a distinct and uniform vertically stressed column. The direction of the gravity force in the ore column is more likely to be transferred along the contact points of the rock chunks, making the force oblique to its original vertical direction in the real underground environment. The lack of this vertical force precludes the effects of the stressed column and would therefore not be a critical issue for design in the actual mine, though it does remain a consideration.
3.8 Summary of Results

The results from the physical model tests indicate:

1. High recoveries are dependent on strong vertical flow. The vertical flow is primarily influenced by steep ring gradients and wide drawpoints.

2. Steep ring gradients required for high recoveries must also be accompanied by large ring burdens. Large burdens prevent the wall waste from entering the ore stream early in the extraction process.

3. Wide drawpoints increase the width of the mass flow zone and improve vertical flow. The wide and strong flow zones are necessary for recovery of the large burdens and increased sublevel intervals.

4. Particle size and uniformity of the ore column and waste rock varied results significantly. Variations to the size of the waste rock changed the flow pattern within the model.

5. A dividing layer existed between moving and static rock. This boundary is referred to as the Sliding Plane (SP) concept in this thesis. The Sliding Plane (SP) within the ore column is conical in form and is represented in Figure 40.

The use of the Sliding Plane concept as a sublevel cave design tool is especially useful because it combines and evaluates the effects of:

- Particle surface friction angle
- Particle intrinsic material properties (density, strength, moisture content)
- Superincumbent stress or Overburden load
- Particle size distribution and uniformity
- Discharge outlet dimensions
These factors are all determinants in establishing the flow behaviour of rock within the stope. The Sliding Plane concept rationalizes the intangible relationship between these factors with regard to their interdependent nature. The use of the SP concept simplifies design and incorporates practical knowledge to the assessment of flow behaviour. There are two critical components to the form and placement of the SP:

1. Effective Digging Depth, and,
2. the Internal Cave Angle.
The effective digging depth controls the horizontal placement of the SP while the average internal cave angle governs its grade. The average of the internal cave angle is referred to as the Sliding Plane Angle (SPA). The value of the SPA ranges between 70° to 80° depending on the size distribution of the caving rock and the width of the drawpoint. The effective digging depth is the average bucket depth into the muckpile during extraction of the ring. This value is typically between one-third to two-thirds of the rings burden. More accurate assessment of the effective digging depth and the SPA can be made according to the recovery values and the source of dilution.

The SP concept would facilitate sublevel cave stope design by inserting the Sliding Plane into a two-dimensional side view of the stope, as was done in Figures 33, 34 and 35. The SP begins at the limit of the effective digging depth and extends upward into the stope area at a grade equal to the SPA. The confluence of the SPA and effective digging depth combine to indicate the initial intersection point of the plane with the burden or Burden Intersection Point (BIP). Optimal designs would set the burden at 30 to 50% greater than the BIP when the ring gradient is equal to that of the Sliding Plane. The ring gradient would be optimized under the stated condition and run parallel to the ore/waste boundary.

Designs which set the burden size larger than the digging depth assume the development of a flow ellipsoid form within the ore column. The evidence of the ellipsoid form was noted in the 4.12 m burdens where wall waste was apparent in all three ring inclinations. Recall from Figure 34 that the ore/waste boundary of the 75° ring runs parallel to the Sliding Plane. The ore buffer zone between the two limits is 1.14 m and remains intact over the height of the entire ore column. In this case the burden is approximately 40% greater than the Burden Intersection Point. Waste from the wall attributed to 20% of the overall dilution, which indicates that the ellipsoid developed past the ore buffer and into the waste on the side.

The complexity associated with determining the amount of the impingement and the inconsistent results found in the tests make it difficult to quantify the amount of the flow ellipsoid’s impingement beyond the Sliding Plane. The wall waste that was recovered from the 4.12 m burden inclined at 75° and was typically small in size and flaky. This indicates that the ellipsoid did not forcibly gouge into the wall waste area but rather scraped along the ore/waste boundary. This implies that the rounding of the ellipsoid is a consideration for design but is not an overbearing factor. Accentuating on the degree of inflection of the flow ellipsoid serves only
to distract the focus of the design engineer from the most important task which is to improve recovery.

The increased sublevel heights require strong vertical flow within the ore column. The physical models indicate that high recoveries are linked to recovering the ore from the upper apex area. Promoting vertical flow is accomplished by:

- Inclining the ring to near vertical angles
- Proper fragmentation of the ore column so fragments are finely blasted and uniform
- Ensuring that the Burden Intersection Point (BIP) encompasses at least 50% of the burden
- Increasing the drawpoint width

The flow strength along the long axis of the stope is crucial to high recovery. This point will prove to be even more important in the actual mine environment where rock size and uniformity become difficult to control. The size and heterogeneity of the rock in the mine setting will deflect the vertical force through the variable contact points of the caving rock assemblage, thereby significantly reducing the strength of vertical flow.

Conclusions from the physical tests recommend the idea of inserting a Sliding Plane into a two-dimensional side view of the proposed sublevel cave configuration, as shown in Figures 33, 34 and 35. Regard must be taken as to the Burden Intersection Point (BIP) and the Plane Intersection Point (PIP). The former must encompass at least 50% of the ring while the latter must be at a vertical level equal to at least 50% of the ring’s height. Best results are obtained typically when these intersection points cover approximately 75% to 85% of their respective baselines.

The ideas behind the Sliding Plane concept are relatively straightforward and the calculation of an efficient stope design is made possible. The Sliding Plane concept is a practical evaluation of the complex relations between rock flow and geometry within the sublevel cave stope. The successful application of the hypothesis to sublevel cave design is highly dependent upon the quality of the input parameters. Determining the internal cave angle and effective digging depth are important, as these factors provide the support for the application of the concept. Assessment of these two factors must consider the network of influences which affect their value. Practical observation and review of recovery values and dilution sources will provide sound evaluation of these factors.
4.0 EVALUATION OF SUBLEVEL CAVE MINE OPERATIONS

The results and observations for this section are obtained from INCO’s Stobie Mine. Stobie Mine is located in the municipality of Sudbury, on the southern side of the Sudbury Basin. Stobie’s annual production is approximately 3.5 million tonnes of ore, 70% of which is recovered by sublevel cave methods. This method predominates in areas above the 2200 level while Vertical Retreat Mining (VRM) is used below this level. Reserves at Stobie are approximately 90 million tonnes.

Two sublevel cave cross-cuts with 31 metre sublevel intervals were observed between January and March of 1996 in order to determine the layout, mining practices and the flow of the rock in the sublevel cave stopes. These cross-cuts, numbered 2940 and 3020, were both on the 2100 level of Stobie Mine. In January of 1996, observations were made of the 21 metre sublevel interval sublevel stope areas on the 1500 level. Video tape footage was used to record the mucking operations and subsequent ore flow during two, eight hour shifts on the 3020 cross-cut. Also during this period, Dyno Nobel Limited had set-up lights and a fragmentation laser camera at the 3020 cross-cut in order to conduct a fragmentation survey.

The initial recovery values in cross-cuts 2940 and 3020 were approximately 20% lower than anticipated. The 31 metre sublevel intervals have presented challenges to the loading, blasting, drilling and mucking operations. Specifically, the proper blast pattern is critical to the results of the 31 m sublevel stopes and the plan for these has yet to be fully developed. Also, the 12.19 m spacing of the drawpoints on the upper sublevels for cross-cuts 2940 and 3020 present a design change which negatively impacts recovery. Further modifications to operational procedures are expected to increase recovery as experience is gained with the 31 m sublevel cave stope intervals.
4.1 Sublevel Cave Stope Set-Up

The typical layout for a sublevel cave stope at Stobie Mine can be described as:

**21 m Layout**

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sublevel Interval</td>
<td>21.34 m</td>
</tr>
<tr>
<td>Drawpoint Width</td>
<td>4.42 m</td>
</tr>
<tr>
<td>Drawpoint Height</td>
<td>4.27 m</td>
</tr>
<tr>
<td>Drawpoint Spacing</td>
<td>12.19 m</td>
</tr>
<tr>
<td>Ring Gradient</td>
<td>80°</td>
</tr>
<tr>
<td>Ring Burden</td>
<td>2.74 m</td>
</tr>
</tbody>
</table>

To accommodate the increased sublevel interval, the new designs will be modified as follows:

**31 m Layout**

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sublevel Interval</td>
<td>30.48 m</td>
</tr>
<tr>
<td>Drawpoint Width</td>
<td>5.49 m</td>
</tr>
<tr>
<td>Drawpoint Height</td>
<td>4.88 m</td>
</tr>
<tr>
<td>Drawpoint Spacing</td>
<td>15.24 m</td>
</tr>
<tr>
<td>Ring Gradient</td>
<td>80°</td>
</tr>
<tr>
<td>Ring Burden</td>
<td>2.74 m</td>
</tr>
</tbody>
</table>

From the 21 m layout, 7.3% of the stope’s ore was recovered by development drifting while the 31 m layout reduced this figure to 5.8%. The dimensions of the drawpoint opening also affect the amount of the stope which may be recovered free of dilution from the sublevel cave stope. The drawpoint opening provides an immediate void space for the blasted ring. Assuming a 50% swell factor for the blasted ore, the 4.42 m × 4.27 m (19 m²) drawpoint opening in the 21 m layout will induce caving in 37% of the stope upon charge initiation. The 5.49 m × 4.88 m (27 m²) drawpoints in the 31 m layout will induce caving in 29% of the stope, considering the same swell factor. Assuming a 30% swell factor, the values of dilution-free ore for each stope increase to 53% and 41% respectively for the 21 and 31 m layouts. The aspect of recovery of dilution-free ore was important to design engineers at LKAB’s Kiruna Mine and influenced drawpoint dimensions (Hustrulid, A., 1995).

In order to increase the percent of dilution free ore for the 31 m layouts to the same amount as its 21 m counterpart, the dimensions of the drawpoint heading would have to increase to 35 square metres. A drawpoint 7 m wide by 5 m high would be more suited to the 31 m layout and would increase the amount of pure ore recovered per stope. Calculations for the amounts of dilution free ore are found in Appendix 3 – Dilution-Free Calculations.
The amount of dilution free ore recoverable per stope can further be increased by:

1. Increasing the area of the drawpoint opening
2. Maintaining a minimum 3 metre ring spacing
3. Maintaining a gap between the ore ring and the waste. This is maintained by compaction of the waste through proper fragmentation
4. Increasing the angle of the muck on the drawpoint floor prior to charge initiation

The 2940 and 3020 cross-cuts were the first 31 m layouts on the 2100 level. Accordingly, their patterns were not to the exact specifications described above, specifically the drawpoints were 7.5 m wide by 6.1 metres high. The larger drawpoints reflect a conservative design approach in order to ensure movement within the stope. The study conducted by Yenge (1981) agrees with this rationale. A significant point of this research states that the drawpoint width-to-rock size ratio between 8:1 and 38:1 are the minimum and maximum limits for the onset of mass flow conditions. Increasing the drawpoint width to increase the drawpoint width-to-rock size ratio will enhance regular flow conditions.

The drawpoint spacing on the 2100 level was 15.24 metres as per the intended design but the spacing on the upper sublevels was only 12.19 metres. The modifications to design demonstrate a transition from an older design to the newer 31 m layout. It is noted that some experimentation occurred in an attempt to improve the recovery and function of the new layouts.
4.2 Mine Operations

Mining operations and practices for the 31 m sublevel cave layouts at Stobie Mine are similar to those used for the 21 m layouts. The increased sublevel interval requires diligence in drilling, loading and blasting practices. Similarly, the wider drift widths require methodical mucking techniques.

4.2.1 Drilling

The sublevel cave stopes are drilled by a Tamrock Data Solo which is an electric-hydraulic drill with a tube-rod carousel. The tube-rod drilling system greatly improves drilling accuracy and drill hole deviation is reduced to a maximum of 2 percent. The improvements in the drilling technology have made 31 m sublevel interval layouts possible. The maximum hole length in the 31 m layouts can be up to 43 metres, with the hole diameters being set to 115 mm.

The drill is set-up in the centre of the drawpoint and drills 12 to 18 holes at an angle of 10° forward from vertical in a set pattern. The ring pattern for the 31 m layout is set at 2.74 m × 2.74 m (9' × 9'), where the first figure indicates the burden and the second indicates the toe spacing. The weight of the tubes may present problems in the long holes since percussion may be significantly decreased.

4.2.2 Loading & Blasting

The blast holes are loaded with an explosive slurry (IREGEL-RUS emulsion with a density of 1.24 g/cm³) according to a blast plan. The explosive column is at least 3 m away from the collar of the hole. Five rings are loaded at one time and the rings are sequentially primed and initiated. This practice requires that collar primers are used on the rings. This presents a significant disadvantage since the collar prime is being relied upon to initiate the entire column, which may be as long as 40 metres, resulting in poor fragmentation. Although a combination of toe and collar primes is preferable, Ontario labour laws prevent working below primed holes, and holes are considered primed with the placement of the toe primers. The collar primers are placed between 4.5 to 6 metres into the explosive slurry.
Historically, rings have been pre-loaded in order to increase loading efficiency. The increased length of the explosive column in the 31 m layout may require termination of this practice if poor fragmentation persists. Proper priming methods are crucial to complete initiation of the explosive column and they outweigh the benefits of pre-loading. A toe and a collar primer could be used provided that the ring is primed and fired immediately after loading.

Bump cord primer has been tested in the past. Results from this initiation method proved largely unsuccessful and did not provide any advantages over collar primers (Stewart, 1996). The objective of any priming method is to ensure full initiation of the explosive charge. Longer explosive columns present difficulties in this regard and must be given consideration.

4.2.3 Mucking

The drawpoints are mucked by a 6 m³ scoop tram. The width of the bucket is 2.5 metres or approximately one-third of the size of the drawpoint. This requires that the scoop tram extract a bucket from each of the left, centre and right hand side. This pattern causes the centre section to be over-pulled relative to the sides, but this is not considered problematic due to the need to induce flow along the centre of the stope. However, excessive drawing in any section, particularly the sides, can lead to early dilution.

The scoop tram operators are limited to mucking ore that is no greater than 76 cm in all dimensions. The reason for this is the limiting size of the ore pass mantle which is 76 cm in diameter. Ore chunks greater than this size require secondary blasting.

Operators prefer to muck areas of finely blasted ore since it does not slow down the extraction operation, even if this involves moving to another drawpoint. The operators are required to achieve certain production limits per shift and waste and oversize slow down the rate of recovery. Oversize rock also creates unsafe draw conditions by causing hang-ups in the stope. Hang-ups require special care in mucking or in severe situations, may even require re-blasting the brow. Similarly, priming adjacent rings beside a hang-up is dangerous.
4.2.4 Dilution Control

The sublevel cave stopes at Stobie Mine are graded visually by mine personnel. Geologists appraise the drawpoint at least twice per 8 hour shift and grade the muck by size fraction and ore content. The ore has a high proportion of pyrite and chalcopryrite and exhibits a shiny copper lustre (Figure 41). The cut-off level for the stope was considered achieved when the drawpoint reached a 30% instantaneous dilution amount.

Figure 41 – Fragmented Muckpile
The operators are also responsible for evaluating the ore but they rely primarily on the judgement made by the geologists. The primary objective of the operators is to meet production quotas and they are more concerned with the amount of oversize. Excessive oversize at a drawpoint will cause the mucking operation at that heading to halt, even if much of the ring is unrecovered or if the oversize is ore. The mine personnel determine when the next ring should be blasted.

Ideal blast conditions allow for the muckpile to be graded based on the size of the ore alone. The ore should be finer than the waste and the target size for the largest dimension of the ore is 46 centimetres. Geologist’s appraisals of the muckpile’s size can be described as follows:

- Fine Size fraction: -20 cm
- Medium Size fraction: 46 cm
- Large Size fraction: +76 cm

The percentage of each size fraction is recorded as well as an evaluation of the grade. With these observations, the geologist then estimates the number of buckets to be removed from the stope. The stope is mucked in this fashion until all the ore is removed from the drawpoint.

The main type of dilution at Stobie Mine consists of greenstone, low grade quartz diorite and granite all of which originate from the pit or wall area. Dilution was typically in the form of oversize which has to be removed from the cross-cut and blasted to be moved to the waste pass.
4.3 Mine Observations

Blast holes were often not collared according to the blast layout. The outside holes in particular were too far in from the sidewalls, reducing the effective size of the drawpoint opening. A higher concentration of the holes was found in the centre of the drawpoint back. The side holes were drilled at 55° and the internal cave angle was 80 degrees. The test stopes also showed many cases of re-drilled holes.

Loading the long up-holes was a difficult task. It was difficult to ensure continuous column under the conditions in the 31 m sublevel cave layout. The weight of the feeder hose and the explosive slurry up 43 m in the hole required great care and diligence in loading. It was important to try to minimize the number of gaps within the holes.

The results from blasting were varied and were dependent upon the diligence in loading and initiation. The initial muck at the drawpoint was well fragmented. After the initial ore, the size range of the muck was non-uniform and quite varied. Large ore pieces appeared at a point where 30% of the expected buckets were extracted. This ore is believed to be from the toe area of the ring and presumably from the frontal section. The largest ore piece observed during the study was 2 m × 2 m × 2.5 m in size (Figure 42). Excessive oversize would cause the muckpile to be abandoned and operations would proceed to initiate the adjacent ring, regardless of recovery.

The muckpile in the drawpoint had an angle of repose between 40 to 60 degrees. The low angles of repose prevented the operators from digging deep into the muckpile. The muckpile would retain its angle of repose providing that there were no obstructions or blockages. Blockages at the brow were typical and caused the muckpile angle to steepen to 80 or 85 degrees. Mucking with the ore pile at this angle would eventually under-cut the blockage and create a surge in the flow. This surge would flood the drawpoint with rock and cause the muckpile to revert to its previous angle of repose. This cycle continued during the normal course of operations. The average digging depth for the test stopes was approximately 1.8 metres.

The blockages and hang-ups which occurred regularly controlled the flow of material to the drawpoint. The draw of ore would occur as surges, with the muckpile being in a stable state or in a state of sudden flow. Regular flow conditions would occur only around the blockages and this represented a minor component of the movement within the draw channel. Avoidance of the
hung-up sections created uneven draw conditions and disrupted the flow of muck across the drawpoint, leading to premature dilution.

Figure 42 – Oversize Ore

After the surge of muck to the drawpoint, the complexion of the muckpile changed. The fine size fraction evident with the muckpile at 80° was covered over by the large rock fragments that had been caught at the brow. The surges fill the void at the base of the steep muckpiles and tend to place the oversize at the foot. This makes mucking more difficult and time consuming. Large hang-ups must be promptly dislodged in order to maintain even flow.

Approximately 85% of the oversize in the test stopes were ore fragments. These large fragments appeared as early as 20% into the expected pull of the sublevel and they were the main size fraction when the expected pull reached 50 percent. This indicated that poor fragmentation
was occurring near the collar of the blast holes. Further proof of poor fragmentation was in the form of the un-initiated explosive slurry which was present in the large chunks at the drawpoint. An explosive column as long as 1.2 m was found in one such oversized ore piece. In extreme cases, some drill holes maintained a whole barrel shape and did not cause any fragmentation to the surrounding rock.

The dilution source in the test cross-cuts was primarily from the wall rock area. This was determined by the fact that most of the waste pieces were relatively un-oxidized and could be classified as medium in size. Also, waste became apparent at early stages of the pull, which indicated that it had to be from a nearby source. The waste from the pit was rarely seen at the drawpoint because the oversize ore usually halted the mucking operation before the ring could be extracted to that level.

The assessment of the ore’s source and grade was difficult to determine and subject to error. The source of the ore in the sublevel cave mine is difficult to determine due to the surge effect and uneven draw conditions at the drawpoint. The surge creates a sudden mass flow which tends to draw muck from the back of the ring. This movement in effect removes muck from the area of the previously fired ring and brings it to the drawpoint. Since the muck near the drawpoint from this ring is at a 30% instantaneous dilution level, it may pass off as ore. These surges and uneven draw conditions will ‘steal’ the ore from the previous ring which is then considered as recovery for the current ring. This creates false recovery evaluations. Diluted muck from the upper sublevels, which has been caved, may also enter the drawpoint and present muck which is similarly diluted to that found in the previous ring. The low grades associated with this mining method only further compounds this problem as the distinction between waste and ore becomes less evident.

Benches are large sections of the ring which remain un-blasted due to initiation or loading problems. They present losses to the recovery of the stope since these sections do not cave to the drawpoint. Benches may also be sections of highly oxidized muck which have become stable arches within the ore column. Benches present operational inefficiencies which are handled in the following manner:
1. The misfired holes are re-primed in an attempt to initiate the remnant slurry in the explosive column.

2. The bench section is 'bombed' when a burlap bag filled with explosive is wedged into the bench. The explosive charge is then initiated to fragment the bench.

3. Holes are re-drilled from the adjacent ring and angled into the bench. These new holes are then loaded and fired in order to re-blast the bench section.

Benches were common in the test stopes. Three of the 7 rings observed contained benched sections. Most of these sections were caused by misfired holes. The cost and loss of productivity presented by benching and poor fragmentation outweigh the advantages of pre-loading holes. Improved coordination of the mine practices are possible and required in order to prime holes at both the toe and the collar. This may not prevent all benching but will reduce its frequency.

Secondary blasting, or 'bombing', at the drawpoint brow is required when hang-ups clog ore flow. The oversize is 'bombed' in an attempt to dislodge the hang-up. This practice degrades the competence of the brow while possibly de-sensitizing the adjacent loaded blastholes. The secondary blast is particularly important when it is early in the draw of the stope and finer ore is believed to be behind the hang-up. The test stopes had much oversized ore which often followed ore of increasingly larger sizes. Poor fragmentation and long delays between mucking the blasted stope increase the likelihood of hang-ups.
4.4 **Mine Results**

The average recovery for the 21 m sublevel cave layouts at Stobie Mine is 78 percent. The average recovery for cross-cuts 2940 and 3020 is approximately 65% and represent a 20% drop from the 21 m layout. The blast pattern used in both cases was $2.74 \text{ m} \times 2.74 \text{ m}$ with an $80^\circ$ ring gradient. Recall that the upper drawpoint spacing for cross-cuts 2940 and 3020 are less than prescribed by the 31 m layout design. The dilution for the mine is approximately 20% and was not considered in the recovery values since it expected to be the same for both layouts. The 20% dilution amount was confirmed on the 31 m layouts by a study completed by Dyno Nobel Ltd.

The recovery between the layouts is based on the total ore recovered as a proportion of the total ore blasted for that ring. The recovery values for cross-cuts 2940 and 3020 are shown in Figures 43 and 44 respectively.

![Graph showing recovery values for cross-cuts 2940 and 3020](image)

*Figure 43 – 2940 X-Cut Recovery*
The median recovery for the 2940 cross-cut was 63.8% while that of 3020 cross-cut was 60.5 percent. The median values are more representative of the true recovery from the test cross-cuts. The lowest values in the data sets for 2940 and 3020 cross-cut are due to a high proportion of oversize ore in the drawpoint which prevented further mucking. The high values for the two data sets represent a situation of over-drawing. Over-drawing occurs when muck extracted does not originate from the blasted ring. In this situation muck is being removed from the adjacent ring while it is being credited as ore from the currently blasted ring. In extreme situations in the 21 m layouts, overdraw could show 140% recovery for an individual ring. Typically, however, adjacent rings resulted in below average recoveries.

In the mine environment it was difficult to assess the origins of the ore. The well fragmented ore was typically from the blasted ring but some of this muck could have come from the previous ring. The hang-ups, irregular flow, poor fragmentation and low recoveries imply that there is a significant amount of ore left in the previous ring. If large boulders prevent the flow of fine ore located at a higher elevation in the stope before mucking for that ring stops, it is then possible that this ore is drawn and recovered by the flow from the next ring.
The inability to tell exactly where the ore was coming from makes it difficult to evaluate the ‘true’ recovery value for the ring. The low, 30% instantaneous dilution cut-off grade also makes it difficult to tell if the muck is coming from sections of the previous ring designated as waste. Grading of the drawpoint involved estimation by the geologists and by the operators. The geologists are concerned primarily with the metal content of the drawpoint but their involvement is limited to a maximum of 2 visits per shift. The operators see the full development of the drawpoint but their objectives are focused on meeting production targets. The objectives from the two groups of mine personnel are not always in agreement.

4.4.1 Ore Column Fragmentation and Recovery

During the period of the underground study, the explosives company, Dyno Nobel Ltd. conducted fragmentation studies on the sublevel cave stopes. Their study examined the effects of different blast designs upon the size and gradation of the blasted ore column. The fragmentation surveys used an electronic camera placed on the back of the drawpoint which recorded the size distribution of the muckpile. The purpose of the study was to find a blast pattern which would minimize benching and improve ore flow in the sublevel cave stopes.

The study indicated that the amount of oversize present in the draw decreases the recovery as illustrated in Figure 45. This graph is a trend line from data obtained from 2940 and 3020 cross-cuts, comparing fragmentation to recovery. The graph shows that increased amounts of oversize in the ring decreased recovery, proving the importance of proper fragmentation for sublevel cave operations. Recoveries could be less than 60% if the ring is composed of 30% oversize. The oversize in the drawpoint reduced the mobility of the ore, thus preventing high recovery.

Blasting efficiency is evaluated by measuring the percentage of muck which is able to pass the designated blast size. The target ore size at Stobie Mine is 25 cm. Therefore, superior blast patterns pass a higher percentage of ore below the 25 cm limit.
In January 1993, Stobie Mine changed their blast patterns to 2.44 m burden by 2.74 m toe spacing (or 2.44 m x 2.74 m) with 115 mm (4.5") diameter holes. Fragmentation results from this pattern yielded 66% passing 25 cm with powder factors between 0.83 kg/t and 0.99 kg/t. Hole sizes were then changed from 115 mm to 102 mm (4.0") with the same blast pattern. Fragmentation then increased to 74% passing 25 cm while powder factors decreased to between 0.70 kg/t and 0.96 kg/t.

The most recent fragmentation surveys compared 102 mm blast holes with patterns dimensioned: 2.44 m x 2.44 m (8' x 8')
2.74 m x 2.74 m (9' x 9')
2.74 m x 3.05 m (9' x 10')
3.05 m x 3.05 m (10' x 10')

The results from these surveys are shown in Figure 46. The bars on the graph represent the percentage of ore passing 25 cm. The line on the same graph indicates the powder factor for that particular blast pattern.
Figure 46 – Blast Pattern Comparison

The 3.05 m × 3.05 m pattern shows an unacceptably low passing percentage. As a guideline this value should be at least 60%. Although, the powder factor for the 3.05 m × 3.05 m design is ideal, poor fragmentation will lead to low recoveries and make this configuration impractical. The best passing 25 cm results for the study were from the 2.44 m × 2.44 m pattern with 76%. However, the powder factor for the 2.44 m × 2.44 m was too high at 0.97 kg/t, as acceptable ranges are between 0.50 kg/t and 0.70 kg/t. The results from this study imply that the 2.74 m × 3.05 m pattern falls within reasonable limits for percent passing 25 cm and powder factors.

The passing value for the 2.74 m × 3.05 m pattern was 66%, only 1 percentage point less than that of the 2.74 m × 2.74 m blast pattern. The powder factor between the two blast designs decreased significantly from 0.74 kg/t to 0.56 kg/t with the 2.74 m × 3.05 m pattern; representing a 32% reduction.
4.5 Summary of the Sublevel Cave Mine Operations

The most significant points evident from the underground operations can be classified as follows:

1. Fragmentation

Fragmentation is key to the sublevel cave mine. Proper fragmentation of the ore column results in improving the efficiency of all the other operations in the mine. Therefore, it is imperative to implement practices which increase thorough fragmentation. The most obvious practice being the simultaneous use of toe and collar primers. Any initiation method which ignites more of the explosive column should also be studied.

The blast designs should not increase the hole diameters above 102 mm. This size of blasthole is well suited to the 2.74 m burdens. The advantages of the 102 mm holes over larger ones can be summarized as follows:

- Provides good fragmentation results with low powder factors.
- Initiation of the column is faster, increasing the likelihood of igniting the entire column of explosive.
- The tonnage of ore per hole is reduced, thereby reducing the load which must be broken by each hole. This in turn, increases the metres drilled per tonne. However note that reducing the load on each blasthole is more advantageous due to the difficulties in ensuring initiation of the explosive column.

Effective blast patterns also take into consideration the amount of benching. In Stobie Mine, excessive benching was the primary motivation for experimenting with variable blast plans and hole diameters. In this regard, the 102 mm holes proved to be more effective at decreasing benching. High powder factors are also required to reduce benching but may create excessive blast damage and over-break. Benching problems remain unresolved and practical solutions must consider; the blast pattern, blasthole size, primer sequencing, powder factor and degree of fragmentation in order to be efficient.
2. **Loading and Explosive Initiation**

   The process of pre-loading 5 rings should be abandoned in favour of loading one ring at a time and then initiating it immediately. This change would allow the use of multiple primers which would greatly improve the likelihood of firing the entire explosive column. This will improve fragmentation and improve ore flow and recoveries. At least two primers per hole should be used and will make the blast results more reliable.

   The writer believes that more complete explosive initiation will greatly improve fragmentation results. This point becomes apparent due to the high powder factors used in the sublevel cave stopes. These powder factors are more than adequate to break the discussed blast patterns, especially when these patterns are compared to other mining methods. The lack of void space will always tend to increase powder factors in the sublevel cave stope, however lack of initiation means that powder is being wasted. The un-initiated powder does not contribute to the explosive energy of the blast but does add to the amount of explosive used per hole. Efficient blasts initiate the entire column and release all of its explosive energy. Observations underground showed multiple sections of the blasthole with un-initiated explosive. The longest of these observed sections was 1 m. Presumably, most of the remnant explosive is from the toe section of the hole.

   More complete use of the explosive will improve fragmentation results, reduce benching and increase the size potential for the blast pattern. This would lead to efficient blasting of the 3.05 m burdens and make it possible for larger burdens to be contemplated in design.

3. **Drawpoints**

   The drawpoint width must be greater than its height. The large drawpoint width forms a wide band of vertical flow within the stope. The strong flow in the central channel will increase the chance of recovering the upper apex which will improve recovery. Large drawpoints improve fragmentation for the bottom section of the ring since they offer a direct void space. Drawpoint area should be 35 m² with the 31 m sublevel intervals since increasing the amount of dilution-free ore is important to stope design.
The wide drawpoints reduce the potential for hang-ups which constrict and cause non-uniform flow. These drawpoints also compensate for poor fragmentation since the larger muck is able to pass to the floor of the extraction heading. This prolongs the extraction period for the ring which may have ended due to excessive oversize or hang-ups.

4. **Grade Control**

The muckpile should be closely monitored for ore content and grade. The grade is estimated visually by geologists who are not always at the drawpoint. Their estimate is largely based on experience and evaluation of the fragmentation of the muck in the drawpoint. The evaluation is dependent on proper fragmentation of the ore ring, which, as indicated, may be inconsistent. Furthermore, this estimate will be considered valid during the period of one shift.

The origins of the ore reaching the drawpoint are difficult to surmise. Irregular flow, caused by hang-ups, can deviate the draw channel into the region of the previous ring. The rock in this may be well fragmented and it appears as ore to the untrained eye. This problem is further compounded by the low grade of ore being caved and the low dilution cut-off point used in the mine. Operator training and awareness in grade control would improve bucket quality.

5. **Cycle of Operations**

Careful scheduling of mine operations can be used to improve fragmentation and recovery. Fragmentation can be improved by timing operations so that explosive columns contain more than one primer. This would involve loading one ring, firing the explosive and this would be followed by mucking. The precise schedule would depend on the availability of personnel, equipment and the number of open drawpoints.

Hang-ups at the throat of the drawpoint should be removed promptly since they affect flow conditions and cause uneven draw. To minimize its influence, the hang-up should be undercut and removed from the draw channel while larger hang-ups may require secondary blasting. In any case, the obstruction should be removed within 15 buckets of the appearance of the hang-up. Hang-ups tend to increase the flow toward the back of the muckpile; this increases dilution and retrieves rock from the region of the previously fired ring.
5.0 CONCLUSION

The advent of improved drilling technology has made increased sublevel intervals possible in the sublevel cave mine. Hydraulic tube rod drill rigs have made long up-holes accurate with less than 2% deviation. The 31 m layout strains all of the mine's operations; the most notable problems from this study lie with loading and explosive initiation. The successful application of the 31 m sublevels will depend upon achieving a high degree of success within each individual operation. Thorough fragmentation of the ore column is critical to this success due to its impact upon all operations.

Lack of proper fragmentation of the ore column will make 31 m sublevel cave layouts impractical. Savings to development costs and economies of scale will be overshadowed by recovery losses and dilution. The steps required to ensure proper fragmentation of the 31 m layouts must be taken in order for the recoveries to be comparable to their 21 m layout counterpart.

Observations from this study advise against the practice of pre-loading blast holes. The advantage of pre-loading offers little benefit when compared to the disadvantage it places upon explosive initiation. The barrier to single ring loading conflicts with the mentality of placing a high quantity of explosive per shift with little thought given to the quality of initiation. This feeling by mine personnel may be overcome if the miners were educated about the implications of thorough fragmentation. Also, improved scheduling of mining operations would lead to loading multiple rings in different drawpoints, thereby maintaining a high quantity of explosive loaded per shift.

The results from the physical model indicate that 31 m sublevel intervals are acceptable in the sublevel cave stope. In fact, the highest test recovery for the 31 m sublevel interval was greater than the highest recovery at the 21 m sublevel interval. The test recovery values for the 31 m sublevel intervals significantly improved when the drawpoint width was increased from 4.42 m to 5.49 m. Mine and model observations indicate that the size of the drawpoint dictates the width of mass flow zone in the centre of the stope. The core mass flow zone is critical to maintaining particle mobility within the stope and must be large enough to accommodate the large ore fragments being drawn. Yenge (1981) suggested that mass flow conditions would definitely occur at a drawpoint width-to-rock size ratio of about 38:1. This condition is not
obtainable in the mine environment due to the overly large openings required and subsequent stability problems. Even though mass flow conditions are not obtainable, increasing the drawpoint width-to-rock size ratio improves ore flow. Draw irregularities caused by hang-ups, particle interlock and un-even draw can similarly be improved as the width to rock ratio is increased.

The wider drawpoints also increased the internal cave angle of the rock during flow. This was one of the most significant observations from the model due to its importance underground. The internal cave angle is largely controlled by the frictional properties of the caving material. The similarity of the physical properties between the model and actual rock is high, as are their respective frictional values. Friction of the material is not affected by the difference of scale between the model and actual stope areas. This makes it likely that the effects found in the model will also occur underground.

The internal cave angle varied only according to the width of the drawpoint. The other geometrical parameters did not affect its value. The internal cave angle varied during the withdrawal of rock from the drawpoint opening. The caving angle reached a stable limit when steady state draw conditions were achieved, denoting the full radial development of the draw cones. The boundary between the stable draw cone and the static rock is referred to in this thesis as the Sliding Plane. The Sliding Plane concept was developed primarily from model results. When applying this concept to the underground sublevel cave mine, one must consider the model’s deficiencies and differences. Nevertheless, it can be used as an effective design tool due to the frictional link between model and actual space.

The Sliding Plane concept is developed in Section 3.7 and 3.8 of this thesis. The concept can be applied to sublevel cave stope designs by drawing a two dimensional side view of a drawpoint area, as demonstrated in Figures 33, 34 and 35. Use of the SP design tool requires the following procedure:

i) Determine the Sliding Plane Angle (SPA). Model results indicate values would range between 70° and 80°.
ii) Determine the effective digging depth of the machinery. Underground observation indicate that this value ranges between one-third and two-thirds of the burden size.

iii) Draw the Sliding Plane (SP) from the limit of the effective digging depth at the drawpoint floor. This line is drawn upward toward the caved area of the stope.

iv) Set the ring gradient and burden size to meet two criteria:

   1. Burden Intersection Point (BIP) $\geq 50\%$ of burden, and,
   2. Plane Intersection Point (PIP) $\geq 50\%$ of ring height.

The physical model represented an idealized flow regime for sublevel cave stopes. The ore within the model was thoroughly fragmented and it displayed a high degree of mobility which would not be common in the mine environment. The high mobility and lack of particle interlock are the main differences between the flow regime in the model and the actual sublevel cave stope. The ideal flow conditions of the model resulted in displaying a sense of proportionality to the geometrical parameters of the layout. It followed that increased sublevel intervals required larger burdens for improved recovery. However, the larger burdens would not be realistic underground due to the low fragmentation, particle mobility and inconsistent blast results.
5.1 **Recommendations**

The integration of concepts developed for sublevel cave mining in this study lead to the following design and operation recommendations for 31 m sublevel intervals:

1. **Drawpoint Dimensions**

   The appropriate dimensions for the drawpoints for sublevel cave stopes should be 5 m high by 7 m wide. The width is the most important dimension due to its control on flow and recovery. The height of the drawpoint should allow for a reasonable height to width ratio and must be able to accommodate drilling and mucking equipment. The large drawpoints increase the number of dilution-free buckets extracted per ring due to the large void area provided for the blast.

   Flow in the sublevel cave stope will be improved by increasing the drawpoint width. The drawpoint width to rock size ratio provides a useful indicator to expected flow. Production blasts attempt to achieve a fragment size of 25 cm; if this size could be consistently maintained, the drawpoint should be at least 9.5 m in order to achieve mass flow based on an assumed ideal drawpoint width-to-rock size ratio of 38:1. Yenge (1981) found that this ratio at 8:1 significantly reduced flow irregularities caused by hang-ups and uneven draw. The width of the drawpoint could be increased but must be kept in check by span stability which should not incur excessive support.

2. **Drawpoint Spacing**

   The drawpoint spacing, assuming a 7 m wide drawpoint, should be at least 12 metres. The range for the spacing could be between 12 m and 24 m, and is dependent upon ore fragmentation and flow. The wider spacings are only possible if near mass flow conditions are maintained during draw.

   The range for the drawpoint is largely based upon model results and the Sliding Plane concept which indicates a stable 80° angle at the onset of steady flow. The Sliding Plane Angle, under the current drawpoint width and spacing, places the boundary along the inner edge of the
upper drawpoint openings. The central channel is then focused on the ore in the upper apex which, as described in this study, is crucial to achieving high recovery.

3. **Blast Pattern**

The appropriate blast pattern is dependent on the results obtained in the underground stope. The results from the Dyno Nobel fragmentation survey indicate that the 2.74 m burden by 3.05 m toe spacing pattern provided efficient blast results. The operations are limited to 2.74 m burdens due to the use of single collar primers which produce inconsistent blast results. Greater efficiency in explosive initiation, using multiple primers or another improved system, would allow for burdens to be increased to between 3 m and 4 m.

Initiation methods warrant further investigation. This would include investigating different primer types and possibly even reducing hole diameters.

4. **Ring Gradient**

The ring gradient should be inclined 80 to 85 degrees toward the caving stope area. The steep gradients are needed to ensure the strong vertical force in the stope. The ring gradient should be increased in relation to the amount of oversize present in the blast. Poor fragmentation may even require that the rings should be inclined vertically although inclining the rings below the vertical is generally preferable in order to maintain brow competence and reduce over-break.

5. **Mine Practices**

The efficiency of the cycle of mine operations can be improved by proper scheduling. This would enable the use of multiple primers in the rings which are loaded, fired and mucked within a short period of time. Removing rock from the stope soon after the blast reduces the amount of oxidation which occurs with sulphide ores.

The development of hang-ups or other obstruction should be promptly removed from the drawpoint. A suitable time frame for removing the hang-up is within 15 buckets of its appearance, this amount representing less than 5% of the total expected draw from the stope. The
use of this guideline will reduce the disturbance caused by the hang-up which will result in improved recovery and reduced dilution.

Operators must attempt to consistently muck deep into the muckpile. The digging depth affects the depth of the draw channel, and is necessary to allow the flow of large ore fragments and for larger burdens. Mucking should cease when rock appears at the throat of the drawpoint. Hang-ups from the next ring will cause the ore on the drawpoint floor to be removed as under-cutting would be required to dislodge the obstruction.

The results of secondary blasts in the throat of the drawpoint should be closely monitored by geologists. The blasts may significantly affect the flow regime and may create excessive dilution. Operators should stop mucking if sudden in-rushes of waste persist during mucking.

6. **Ring Drilling**

Drillers must take extra precaution to follow drill patterns according to plan. Deviations from these plans should be recorded and noted to mine geologists and engineers. A common defect in ring drilling was that the outside holes were not out far enough against the walls. These holes also tended to be at steeper angles than described in plan.

The angle of the side holes should be set to a lower angle than their current $55^\circ$. A lower side hole angle is preferable in order to ensure fragmentation of the apex area between adjacent cross-cuts. The fragmentation is important for the draw of the lower drawpoint and will increase its recovery. The draw conditions of the drilled drawpoint will not be affected because the internal cave angle for the heading is greater than $55^\circ$. 
5.2 Suggested Areas of Further Study

1) Fragmentation studies using multiple primers should be conducted. Different types of primers, such as bump cord, could also be used with the aim of improving initiation efficiency. This type of study could also test the differences between short and long delay primers within the blast plan.

2) Blast patterns could be altered with the aim of reducing benching and oversize. Improved types of explosive or priming methods could enable altering the burden and toe spacings for current blast designs.

3) A cost analysis which compares the operational savings to the recovery loss between the 31 m and 21 m sublevel cave layouts should be conducted. This type of analysis would provide true insight into the economic practicality of the increased sublevel intervals.

4) Studies could be conducted to assess the economic benefit of improving recoveries in comparison to the expense of altering geometric plans. This would include studying the economic gain of increased recovery due to wider drawpoints in comparison to the expense of increasing ground support.

5) Incorporate the design principles of the Sliding Plane concept into a three-dimensional computer simulation which is capable of altering ore sizes and measuring recovery.

6) Construct a physical model, larger than the one from this study and with a reduced scale. Tests would pay particular attention to particle size and uniformity while changing drawpoint width and spacing.
REFERENCES


Kvapil, R., Ore Flow by Block and Sublevel Caving, SU-RP (State Institute for Mining – Prague), 1954. Part I & II.


Preston, C., Dyno Nobel representative, INCO Mines Research, Copper Cliff, 1995, Personal communication.


APPENDIX 1

PHYSICAL MODEL RESULTS

Table #1 – Physical Model Ordered Results

Table #2 – Physical Model Grouped Results
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<th>TOTAL % DILUTION</th>
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<th>% CAP WASTE</th>
<th>% RECOVERY</th>
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APPENDIX 2

DILUTION SOURCE

Table #1 – Dilution Results for the 70’ Sublevel Intervals

Table #2 – Dilution Results for the 100’ Sublevel Intervals
### TABLE #1 - Dilution Results for the 70' Sublevel Intervals

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<th>Wall</th>
<th>Cap</th>
<th>Recovery %</th>
<th>Rank</th>
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### TABLE #2 - Dilution Results for the 100' Sublevel Intervals

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<td>100</td>
</tr>
<tr>
<td>18' - 75°</td>
<td>17,18</td>
<td>73</td>
<td>1</td>
<td>6</td>
<td>94</td>
<td>76</td>
<td>1</td>
<td>6</td>
<td>94</td>
</tr>
<tr>
<td>18' - 80°</td>
<td>28</td>
<td>71</td>
<td>2</td>
<td>45</td>
<td>55</td>
<td>75</td>
<td>2</td>
<td>52</td>
<td>48</td>
</tr>
<tr>
<td>18' - 85°</td>
<td>29</td>
<td>52</td>
<td>8</td>
<td>100</td>
<td>0</td>
<td>61</td>
<td>8</td>
<td>100</td>
<td>0</td>
</tr>
</tbody>
</table>
APPENDIX 3

DILUTION-FREE
CALCULATIONS
% of Development Ore per Stope

1. 21 m Layout

Stope Area = 21.34 m × 12.19 m = 260.14 m²
Drawpoint Area = 4.42 m × 4.27 m = 18.87 m²
% Development Ore per Stope = 7.3%

2. 31 m Layout

Stope Area = 30.48 m × 15.24 m = 464.52 m²
Drawpoint Area = 5.49 m × 4.88 m = 26.79 m²
% Development Ore per Stope = 5.8%

Open Volume available in Drawpoint

** Assume a 40 degree angle of repose for the rock in the drawpoint.

1. 21 m Layout

Drawpoint Open Volume = 2.74 m × 4.42 m × (4.27 m / sin 40°) = 80.45 m³

2. 31 m Layout

Drawpoint Open Volume = 2.74 m × 5.49 m × (4.88 m / sin 40°) = 114.20 m³
**In-Situ Volume affected by Drawpoint Open Volume**

a) Assume 50% swell  

b) Assume 30% swell

1. **21 m Layout**

   a) Ore Volume affected  
   \[= 80.45 \text{ m}^3 + (80.45 \text{ m}^3/0.5) = 241.35 \text{ m}^3\]

   b) Ore Volume affected  
   \[= 80.45 \text{ m}^3 + (80.45 \text{ m}^3/0.3) = 348.62 \text{ m}^3\]

   a) Ore Area affected  
   \[= 241.35 \text{ m}^3 / 2.74 \text{ m (burden)} = 88.1 \text{ m}^2\]

   b) Ore Area affected  
   \[= 348.62 \text{ m}^3 / 2.74 \text{ m (burden)} = 127.2 \text{ m}^2\]

2. **31 m Layout**

   a) Ore Volume affected  
   \[= 114.20 \text{ m}^3 + (114.20 \text{ m}^3/0.5) = 342.60 \text{ m}^3\]

   b) Ore Volume affected  
   \[= 114.20 \text{ m}^3 + (114.20 \text{ m}^3/0.3) = 494.87 \text{ m}^3\]

   a) Ore Area affected  
   \[= 342.60 \text{ m}^3 / 2.74 \text{ m (burden)} = 125.0 \text{ m}^2\]

   b) Ore Area affected  
   \[= 494.87 \text{ m}^3 / 2.74 \text{ m (burden)} = 180.6 \text{ m}^2\]

**In-Situ Ore Area removed by Caving**

a) Assume 50% swell  

b) Assume 30% swell

1. **21 m Layout**

   Stope Area to be removed by immediate caving  
   \[= 260.14 \text{ m}^2 - 18.87 \text{ m}^2 = 241.3 \text{ m}^2\]

   a) % of Stope affected by Drawpoint Opening  
   \[= 88.1 \text{ m}^2 / 241.3 \text{ m}^2 = 36.5\%\]

   b) % of Stope affected by Drawpoint Opening  
   \[= 127.2 \text{ m}^2 / 241.3 \text{ m}^2 = 52.7\%\]

2. **31 m Layout**

   Stope Area to be removed by immediate caving  
   \[= 464.52 \text{ m}^2 - 26.79 \text{ m}^2 = 437.7 \text{ m}^2\]

   a) % of Stope affected by Drawpoint Opening  
   \[= 125.0 \text{ m}^2 / 437.7 \text{ m}^2 = 28.6\%\]

   b) % of Stope affected by Drawpoint Opening  
   \[= 180.6 \text{ m}^2 / 437.7 \text{ m}^2 = 41.3\%\]