Sublevel caving – past and future

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Abstract

The paper reviews the historical development of sublevel caving and presents some ideas concerning its future application and development. The "Present" status is not covered since it will be the topic of a number of other papers offered at the conference. The sublevel caving technique evolved out of top-slicing in the early part of the 20th century. Block caving was a natural outgrowth of sublevel caving. Further development of sublevel caving moved very slowly until the early 1950's when the technique blossomed into today's version, largely due to improvements in drilling technology. In addition to being of historical interest, the development path also has significance for the future development of the sublevel caving system. This will be shown. Although the principal application of the method throughout time has been to the extraction of iron ore, this may be about to change. Many large open pit mines are rapidly reaching their final economic depths and sublevel caving is a natural candidate for underground continuation. For mining companies it is natural to consider the largest scale, most modern version of the method. There are both opportunities and concerns regarding the application of large-scale sublevel caving. These will be discussed. Some innovative, and possibly even implementable, design modifications will be presented.

1 Introduction

The sublevel caving technique according to early mining books (for example Peele (1918)) evolved in the U.S. from top slicing. It was a logical next step in the mine geometry scale-up process. Block caving, in turn, was the logical scale-up from sublevel caving.

Janelid (1972) indicates

In the first application of sublevel caving, the ore was not drilled and blasted completely between two sublevels, but certain parts were broken by induced caving (hence the name sublevel caving). As the method is applied today, the whole quantity of ore between the different sublevels is broken (or at least should be) using controlled drilling and blasting. If this is done in a proper and rational way, there are good possibilities of developing a mining method which can be applied, technically as well as economically, on any orebody of suitable size, location and rock mechanical properties.

In spite of some searching, the modern origins of today's version could not be clearly identified. Possibly it developed in the iron mines of Sweden. Janelid (1972) indicates

"For a long time, sublevel caving was the predominant mining method at Grängesberg. During the last ten years (since about 1960), however, block caving has given 70% of the production."

In 1960, the sublevel caving technique was being used by 19 Swedish mines with a total yearly production of about 9.5 Mton (Ohlsson, 1961). Figure 1 is a sketch of the method as practiced at LKAB's Kiruna mine at about that point in time.

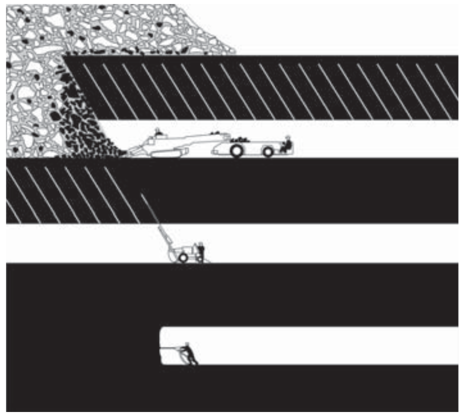

Figure 1 Composite section view of the sublevel caving mine at Kiruna in 1957

The scale was small, certainly by today's standards, with a sublevel spacing of 9m, a drift size of 5m x 3.5m, and a sublevel drift spacing of 10m center-to-center.

As Janelid (1961, 1972) pointed out:

"Sublevel caving is in many respects simple. It can be used in orebodies with very different properties and it is easy to mechanize. However, from other points of view such as recovery, dilution and similar, the method is unfavorable. The designs which are used and the measures which can be taken to eliminate the disadvantages are poorly understood. In the end of the 1950's, model tests regarding gravity flow in material resembling broken rock were started at the Division of Mining, the Royal Institute of Technology (KTH) in Stockholm. The purpose was to study how the geometrical design of various parameters in sublevel caving are influenced by the motion which is induced in the material when ore is loaded in a sublevel drift. Some of these model tests were performed as a part of senior theses and others by assistants and research engineers. Model tests and extensive literature studies on sublevel caving have also been carried out in Kiruna together with conducting practical tests underground. The results achieved have been so encouraging that continued research work is well justified since the economic benefits which can be achieved through the development of the correct method are extraordinarily large."

In Czechoslovakia in 1950, Rudolf Kvapil was given the task of determining the causes of problems in bins and silos and, based on this new understanding, to develop ways of improving their performance. It was evident to him that it would first be necessary to determine the basic gravity flow principles for granular and coarse materials since they must be completely different from principles describing the flow of liquids which were then available for use. He decided that the only realistic way to proceed was to construct and test a large number of models and to make in situ observations. Many of these models and the knowledge gained are described in his recent book (Kvapil, 2004). In 1965, Kvapil joined Janelid at KTH and began applying the gravity flow principles gained in the study of bins and silos to sublevel caving. Figure 2 shows one of the elaborate sand models constructed as part of the studies. With draw, the ore contained within the black ellipsoid of extraction disappears through the drawpoint as the ellipsoid of loosening forms.

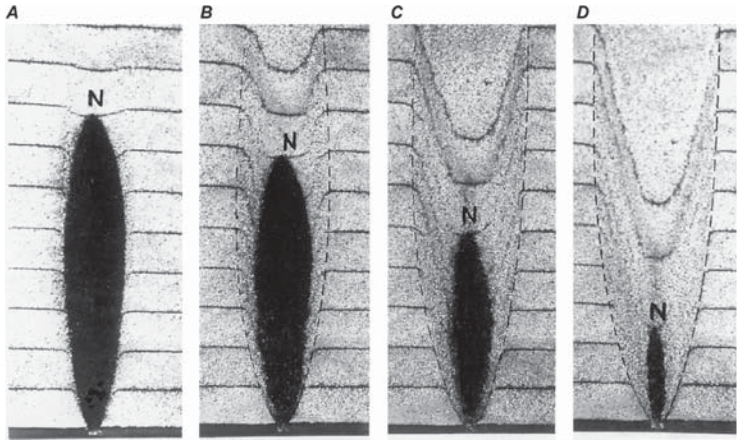

Figure 2 Typical sand model showing the draw ellipse. Kvapil (1965, 1982).

Figure 3 shows the application of this type of model to a sublevel cave design. In this particular case, the sublevel spacing is 12.5m, the drift dimension is 5m x 3.5m, the sublevel drift spacing is 12m and the burden is 2m. These closely resemble the sublevel dimensions used by the Kiruna mine in the early 1980's. Thus model-based approach to developing sublevel caving geometries was very successful.

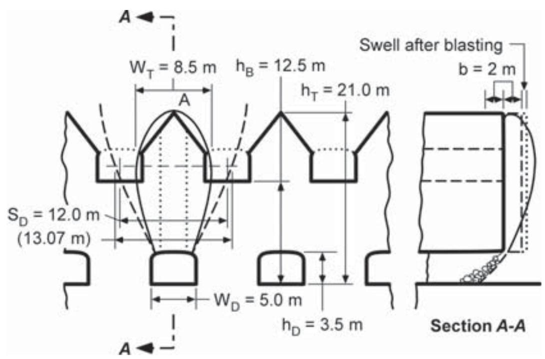

Figure 3 Application of gravity flow principles to sublevel caving design. Kvapil (1982, 1992)

Over the past few years, the scale of sublevel caving has increased markedly with LKAB being a leader in this regard. Figure 4 provides a comparison of the sublevel caving mining geometries appropriate for the years 1963, 1983 and 2003 at the Kiruna mine. Some of the important parameters are tabulated in Table 1.

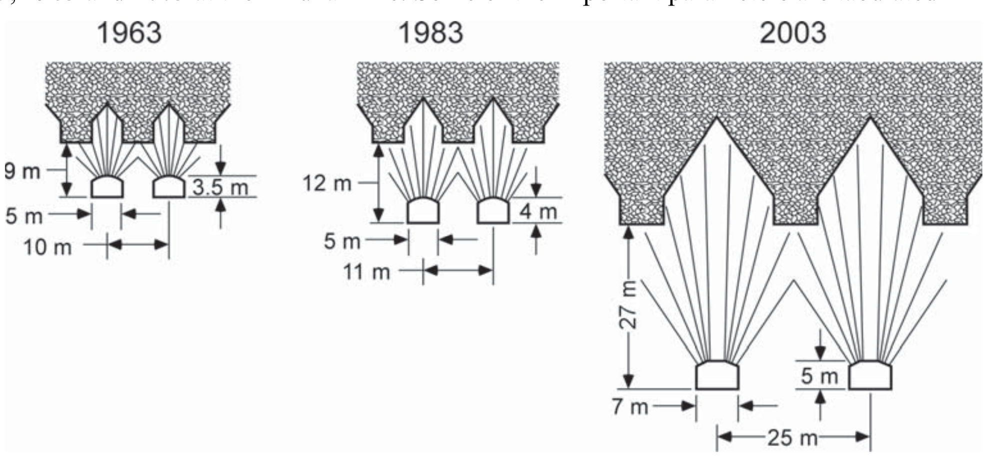

Figure 4 The sublevel caving geometry at the Kiruna mine at three different points in time. Marklund and Hustrulid (1995)

At the Kiruna mine today the sublevel spacing is 28.5m. In certain sectors of LKAB's Malmberget mine, the sublevel spacing is as high as 30m.

Table 1 Summary of some important design parameters (Marklund and Hustrulid, 1995)

	Year		
Parameter	1963	1983	2003
Drift width (m)	5	5	7
Drift height (m)	3.5	4	5
Sublevel height (m)	9	12	27
Sublevel drift spacing (m)	10	11	25
Blasthole diameter (mm)	45	57-76	115
Burden (m)	1.6	1.8	3
Holes/ring	9	9	10
Tons/ring (t)	660	1080	9300
Tons/meter of drift (t/m)	400	600	3100

In 1970, Kvapil left KTH to pursue other opportunities. Although a number of gravity flow studies have been pursued in Sweden and elsewhere since that time, questions still remain concerning the appropriateness of the model results and the design recommendations based upon them. Today, with the continuing push to increase mining scale, a fundamental question is whether the gravity flow principles which have well-served small-scale mine designs can be applied at much larger scales or whether some other approach is required. This paper will provide some thinking in that regard.

2 The proof of the pudding is in the mine

2.1 Introduction

There is an old proverb (ask.yahoo.com) that says "The proof of the pudding is in the eating." It means that the true value or quality of something can only be judged when it is put to use. The meaning is often summed up as "results are what count."

Janelid (1972) indicated

"The results of a large number of model tests have been registered with photographs, films, diagrams, tables, calculations, etc. They have yielded very valuable and consistent information.

However, when calculations as well as model tests are used to describe the actual process in sublevel caving, the uncertainty is such that a closer study of the process in full scale at a mine is justified. Here it is important to clarify the real shape of the volume of motion and the factors in the mining method that have influence on the shape. A comparison between different parameters in model and full-scale is also of great value for following studies."

In short, even at this early stage in modern sublevel caving, there was a concern as to how well the sand models actually represented reality. The proof was in the mine waiting to be demonstrated. Figure 5 presents this thinking in simple terms.

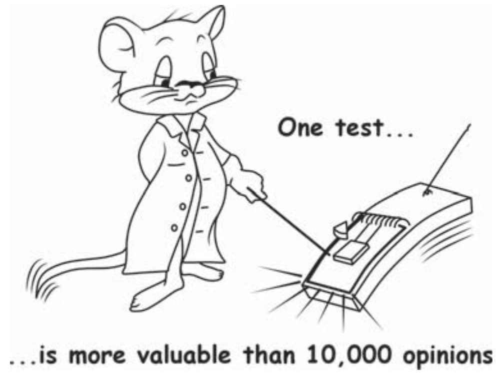

Figure 5 A wise Norwegian saying

The result was the planning and carrying out of the first full-scale test at a mine involving markers in Grängesberg in 1971. Since that time, only a very few others have been conducted. In this paper, the focus will be on the Grängesberg tests and those conducted at the Kiruna and Perseverance mines.

2.2 The Grängesberg mine marker tests (Janelid, 1972)

To respond to this concern, a program was developed to conduct a full-scale test at an iron mine, Timmergruvan, which is located at Grängesberg in central Sweden. Sponsorship was provided by the Swedish Mining Association, Gränges Stål and LKAB. The main goals of the test were:

- Investigation of gravity flow during loading of rock which has been blasted against loose rock
- Investigation of the influence of the scaling factor when comparing model tests and full-scale tests

The main test was carried out between the 322m and the 335m levels of the mine. It involved 3 sublevel drifts on the 335m level and 4 sublevel drifts on the 322m level. Initially the sublevel drifts were driven 3m x 3m with a 4m wide pillar separating the drifts. Those on the 335m level were eventually widened to 3.5m in order to get an adequate size and adapt the drifts to the drilling equipment. The markers were inserted in the rings to be blasted from both the 335m and 322m levels. There were 5 rows of marker holes in each ring. A total of 12,628 markers were inserted from level 322m and 2,022 markers from the 335m level. About 75% of the markers recovered. The ore was a mixture of magnetite and hematite. The angle of internal friction for the ore and caved rock was determined to be 45 to 50^{o}. When blasting against free space, the volume of rock increases by about 60%. When the blasting is done against loose rock, as is the case in sublevel caving, the swelling is reduced. Janelid (1972) notes that according to literature and our own observations, the "confined" swell in the direction perpendicular to the sublevel drift is about 20%.

Table 2 Summary of some important factors concerning the Grängesberg marker tests

Parameter	Value
Sublevel drift spacing (m)	7
Sublevel spacing (m)	13
Hole diameter (mm)	41
Burden (m)	1.5
Sublevel drift width (m)	3.0 slashed to 3.5
Sublevel drift height (m)	3
Front inclination (degrees)	90

The explosive used was Dynalit in two types of cartridges; 36mm in diameter by 1150mm long and 38mm diameter by 500mm long. For the single ring blasts, the delay pattern was the following:

- the two center holes on a 25 ms delay
- the hole to each side on a 37.5 ms delay

- the hole to each side on a 50 ms delay
- the final two holes on a 75 ms delay

The practical charge density was 0.6 – 0.7 kg/m^3 which was higher than normal in order to get good fragmentation. The test involved the firing of 15 rings. Of these, 9 were fired as single rings and 3 as double rings. Adjoining rounds were loaded simultaneously so that the whole mining front could be retreated evenly.

Track-bound, compressed air driven equipment consisting of an Atlas Copco LM 56 (0.56 m^3 capacity) overshot loader and shuttle car type U2N with a 1.1 m body (1.1 m^3 capacity) were used. Loading was (1) carried out in intervals of 5 cars before the equipment was moved to the next drift and (2) distributed over the whole front slope in the drift in order to get an even motion along the whole mining front. The relatively small and light equipment facilitated recovery of the markers in the front and in the car as well as weighing of loaded cars and estimating the waste rock dilution in the loaded rock. During loading of each car, the numbers of the markers found in the front, bucket or car were registered and recorded as belonging to the car which was loaded when they were found. About 75% of all the markers in the test were recovered.

The degree of filling was estimated according to a certain system and the car was weighed with a 5-ton dynamometer located at the orepass. The waste-rock dilution in each loaded car was estimated visually according to a number of interval limits. The fragmentation was observed by manual screening of a number of cars for each round. Other observations in connection with loading concerned hang-ups and the shape of drift walls and the roof next to the front slope.

Figure 6 shows the average volumes of motion as determined using the markers positioned 15 cm in front of the mining front for loaded ore quantities of 400 tons and 600 tons.

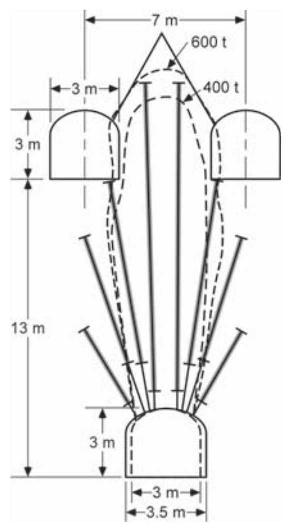

Figure 6 Results of the Grängesberg marker tests. Janelid (1972)

Model tests were carried out at scale 1:20 with a total of more than 15,000 markers installed. The model material was crushed and screened magnetite from Grängesberg and the waste rock was limestone. When packed in the model, the respective densities were 3.24 and 1.6 g/cm^3. Janelid (1972) indicates that

"A comparison between the results from full-scale tests and model tests shows that the shapes of the volumes of motion are somewhat different. In the model tests, the volume of motion is somewhat narrower, higher and more evenly shaped. This is natural with regard to a more evenly sized and more easily moving material in the model. The angles of internal friction have been measured at about 35^o in the model and 45-50^o in normal blasted rock in the mine. In addition, loosening up and density of the material are more favorable for an even gravity flow in the model tests."

Unfortunately, the incursion of dilution as a function of draw was not reported by Janelid (1972) for either the full-scale or the model tests.

2.3 The Kiruna mine marker tests (Quinteiro et al, 2001)

A very detailed marker study was conducted at LKAB's Kiruna mine as part of the "Sublevel Caving 2000" project. The tests were conducted on the 713m level. The insitu density of the magnetite ore is 4.6 t/m^3 whereas that of the waste is about 2.6 t/m^3. Figure 7 shows the fan geometry and Table 3 summarizes some of the important parameters. The length of the longest holes was of the order of 40m.

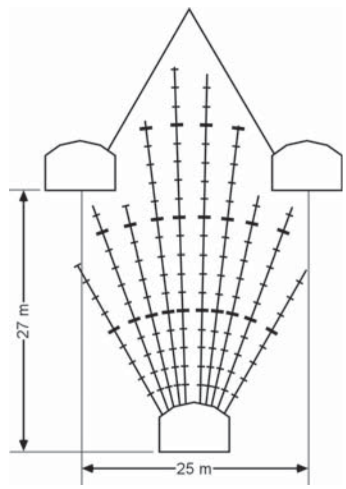

Figure 7 Fan geometry for the Kiruna sublevel cave

Table 3 Summary of some important factors concerning the Kiruna marker tests

Parameter	Value
Sublevel drift spacing (m)	25
Sublevel spacing (m)	27
Hole diameter (mm)	114
Burden (m)	3
Sublevel drift width (m)	7
Sublevel drift height (m)	5
Front inclination (degrees)	80

The emulsion explosive Kimulux R produced by Kimit AB was used. It has a density of 1.2 g/cm3 and a velocity of detonation of 5100 m/s (Nordqvist, 2007). Of the 10 holes in a fan, the middle four holes were shot using short delay intervals followed by the remaining holes.

As noted by Quinteiro et al (2001)

"The sublevel caving layout used at Kiruna has reached dimensions that are far beyond those that formed the basis for the development of the early design guidelines. Thus, there was a need to verify the gravity flow pattern for this very large sublevel caving area. It was decided to install markers in the fans so one could estimate the ellipsoid of extraction."

Figure 8 shows in longitudinal section the 15 marker locations in a fan. The markers were installed in special holes drilled half way between the production rings. A total of 908 markers were installed in 24 fans. Of these, a total of 272 markers were recovered.

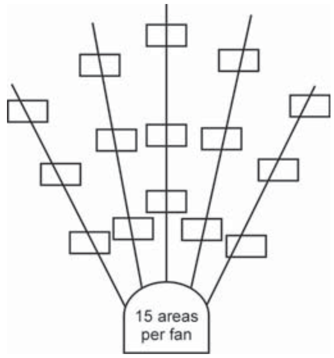

Figure 8 Marker locations in the fan

Figure 9 shows the results of the recovered markers expressed as a percentage of the total number of markers installed at each particular location.

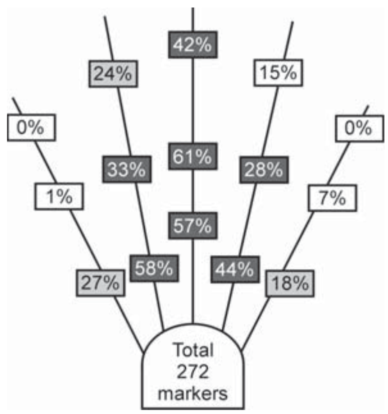

Figure 9 Percentage of the recovered markers at a particular position

It can be seen that only a very small number of markers were recovered from the sides of the fan indicating that the ore flow was small. On the other hand, a large number of markers were recovered from the central part of the fan indicating that the predominant ore flow pattern was in the center. This type of flow behavior will result in early dilution. Figure 10 shows the results in Figure 9 in the form of a contour plot.

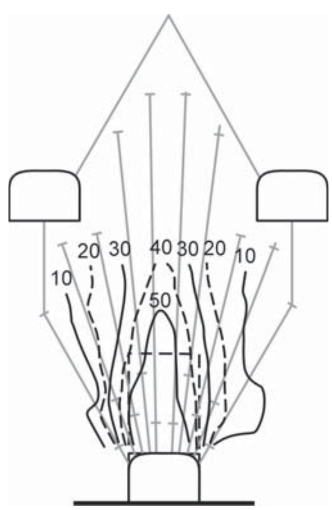

Figure 10 Contour plots showing the percent recoveries at the different marker positions

An extensive sampling program was carried out during the loading out process. The objectives were:

- to gather knowledge about the behavior of waste inflow and changes in ore quality
- to control drawing of the fans with the objective of optimizing ore recovery and separating the different ore qualities
- to test software developed for draw control procedures.

The sampling technique consisted of taking a 1-kg sample for every tenth LHD bucket. Figure 11 shows one example of waste inflow measured by sampling the drawing of a fan.

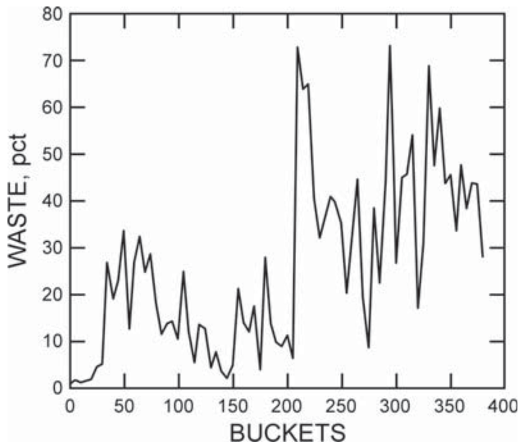

Figure 11 An example of waste pulsation, Quinteiro et al (2001)

As Quinteiro et al (2001) explain

"The data showed important differences in the behavior of waste inflow between laboratory tests and in situ measurements. The usual laboratory behavior of a smooth increase in waste percentage as the fan is drawn is more the exception than the rule in very large sublevel caving. The usual behavior observed is that of a pulsating waste inflow, i.e., high peaks of ore inflow are followed by high peaks of waste inflow. The probable explanation for this behavior is that blasting produces a granular material with widely varying fragmentation sizes and mobility characteristics throughout the fan.

Normal laboratory gravity flow experiments do not account for this factor, i.e., the materials used, such as sand, have the same mobility across the fan. Thus, the pulsating behavior of the waste inflow observed in situ makes the draw control more difficult to be optimized. Procedures have been developed however to follow the drawing of the fans to optimize ore recovery under pulsating waste inflow."

The results achieved in this experimental area of the mine

Ore recovery = 93%

Dilution = 20%

must be considered excellent. However, it should be emphasized that very special draw control procedures were employed.

2.4 The Perseverance mine marker studies. Hollins and Tucker (2004)

The Perserverance underground mine is located 645 km northeast of Perth in Western Australia. It is a disseminated nickel orebody. In 2000 after two major reviews of the sublevel cave performance, several design and operational changes were made. The most notable change was a reduction in the center-to-center sublevel drift spacing from 17.5m to 14.5m with the idea being to facilitate interactive draw. Strict mucking (bogging) practices were also put into effect "in an attempt to avoid surging of waste ingress." The primary design parameters are given in Table 4.

Table 4 Summary of some important factors concerning the Perseverance marker tests

Parameter	Value
Sublevel drift spacing (m)	14.5
Sublevel spacing (m)	25
Hole diameter (mm)	102
Burden (m)	3
Sublevel drift width (m)	5.1
Sublevel drift height (m)	4.8
Front inclination (degrees)	75

The ore density was 3.45 g/cm^3 yielding an approximate ring tonnage of 3500 tons. The draw cutoff grade was 0.9% Ni. The explosive Powerbulk VE having a density of 1.0 g/cm^3 was used. Measurements of fragmentation indicate 90% smaller than 0.4m. No information was provided regarding the firing sequence of the delays.

The marker tests were conducted in five separate crosscuts on three different levels of the mine. A total of 1762 markers were installed at one-meter intervals. By varying the number and pattern of marker rings between blastholes, the depth and width of draw could be studied. The overall flow as determined from the markers is shown in Figure 12.

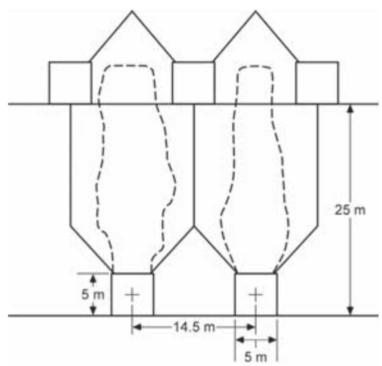

Figure 12 Section showing the rings with the draw pattern superimposed

Their findings are summarized in point form below:

- There was no evidence of interactive draw
- The maximum width of draw measured through the recovered draw markers was 11.5m (+/- 1m). This means a zone of blasted material located between crosscuts and at the toes of blastholes did not report to any of the drawpoints from which the material was fired.
- During the trials, the measured size and width of this zone varied, however generally 35% of the blasted tons did not report to the crosscut from which they were fired or any adjacent crosscuts.
- As more than 100% of design tones were mucked from the marker trial areas, material must have been traveling to the drawpoint from outside the blast envelope.
- To study the depth of draw, up to three marker rings were positioned between each apparent blast burden (3m). On the 9715 level, markers were recovered from positions up to 2 metres in front of the ring being fired.
- The trials indicate that markers placed at the level above can flow into the drawpoint within mucking of 20% of the blasted tons. This is consistent with early observations of barren ultramafic and felsic dilution entry when crosscuts above were mucked to waste.

- Recovery has been from markers positioned directly above the crosscut in an area between the extraction draw zones of adjacent crosscuts on the level above.
- The recovery of markers behind a freshly fired ring is consistent with brow break back records throughout the mine.
- At the time of the writing, 540 markers had been recovered.
- Detailed draw analyses were undertaken for all the marker trial rings. The results showed dilution entry points between 11 and 25% of the tons extracted.
- The analysis of the data confirmed a correlation between hang-ups in the drawpoint and waste surging. From visual observations of the hangups, it appeared that waste egress was coming from the front of the ring as opposed to dilution from above the blasted ring.
- When a hang up did occur, markers from deeper, wider, and lower down in the blast envelope were retrieved.

For the five trial locations, on average 60-70% of the recovered markers were recovered on the level on which they were installed (primary recovery). Additional marker recovery was reported as follows:

- 20 - 25% on the subsequent level (secondary recovery)
- 10 - 15% on the third level (tertiary recovery)
- Up to 8% on the fourth level (quaternary recovery)

2.5 Some summary observations

In reviewing the results of the marker tests from Grängesberg, Kiruna and Perseverance, it is interesting to note that they all basically reveal a type of "silo" flow such as shown in Figure 13 even if the drilling pattern extends far outside of the "silo."

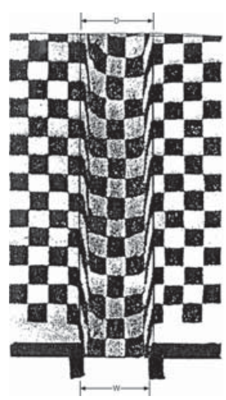

Figure 13 "Silo" type of flow pattern. Kvapil (1955), Janelid and Kapil (1965)

Some "average" primary flow width/drift width ratios (W_f/W_d) for the three cases are summarized in Table 5.

Table 5 A comparison of the marker flow patterns

Mine	Drift width (W_d) (m)	Level interval (m)	Flow width (W_f) (m)	W_f/W_d
Grängesberg	3.5	13	4.9	1.4
Kiruna	7	27	10.3*	1.48
Perseverance	5	25	7.1	1.42

* Arbitrarily taken as the 30% contour

The W_f/W_d ratio of 1.4 – 1.5 seems to apply for small scale sublevel caving geometries as well as very large scale. In retrospect, there are three reasons why this is a very logical result:

1. The middle holes of the ring are fired first and can make first use of the swell volume offered by the underlying sublevel drift.
2. The central holes are drilled sub-vertical, fairly parallel, and relatively close to one another. The result is a relatively high and uniform specific charge compared to the other holes in the round. Thus one would expect the best, most uniform fragmentation
3. The ore material in the central part of the round can make the best use of the effect of gravity in directing it to the drawpoint.

This finding has very important, but unfortunate, consequences to sublevel caving design. The unfortunate part is that the sublevel drift spacing will have to be reduced from the current spacing which is based largely on the dimensions of the draw ellipse. Before suggesting changes, it is important to review the draw mechanics which can explain, at least in part, the marker study observations. These are the subject of the following section.

3 Simplified draw mechanics of sublevel caving

3.1 Introduction

Small scale physical model test results have historically played a very important role in the dimensioning of sublevel caves. In the construction of these models, the sand or other material is simply poured into the forms. As such, the properties are uniform and the mobilities are the same independent of position within the model. In a sublevel cave, this is not the case. All of the material in the fan is drilled and blasted. Because of the fan geometry, the amount of explosive/unit volume and hence the fragmentation varies throughout the fan. The ore material in the center part of the fan and the lower part of the fan has a much higher specific charge than that at the boundaries of the ring. Furthermore, the “cave” which lies in front of the blasted slice is an eclectic mixture of waste rock and ore remnants. Its mobility varies with location and with time (it changes with the extraction geometry). In addition, the sublevel drift which provides the “free” swell is located at the bottom end of the fan. The ore in the near vicinity of the drift has a much greater access to this volume and the chance to bulk. . For these reasons, one might expect a difference in the mechanics of flow between the sand models and reality. The actual mechanics are very complicated and probably impossible to fully describe. In this section, a somewhat simplified description based largely on swell will be presented.

3.2 Swell

3.2.1 Introduction

In modern sublevel caving, the ore in the ring is fragmented by drilling and blasting. The size shape and looseness of the resulting fragments depends upon the blasting design and the size and location of the swell space. If a rock volume is allowed to freely expand when blasted, the resulting broken volume is significantly larger than the initial volume. By knowing the solid and broken densities, one can calculate the percent swell. This has been done in Table 6 for a variety of rock materials.

Table 6 Swell for different materials, Simetric (2007)

Material	Solid Density (kg/m^3)	Broken Density (kg/m^3)	Percent Swell (%)
Basalt	3011	1954	54
Gneiss	2867	1858	54
Granite	2691	1650	63
Limestone	2611	1554	68
Magnetite	5046	3284	54
Marble	2563	1570	63
Porphyry	2547	1650	54

Quartz	2643	1554	70
Shale	2675	1586	69
Stone (common)	2515	1602	57
Trap Rock	2883	1746	65

As shown in Figure 14, in sublevel caving there are two main swell modes for the ore within the slice:

- Downward into the sublevel drift – known as 'free' swell
- Forward against the caved waste – known as 'confined' swell

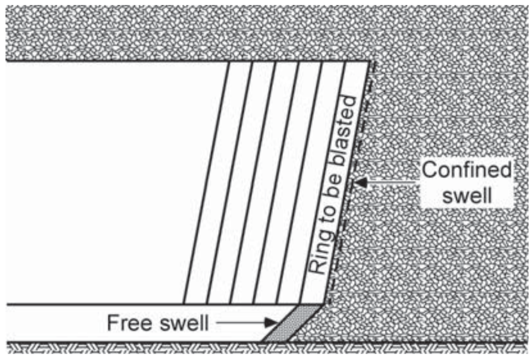

Figure 14 Location of the swell space, Newman et al (2008)

The available "free" swell space as provided by the sublevel drift is highly dependent on mining scale (see Table 7). Furthermore, the proximity of the free swell to the ring boundaries is also highly scale dependent.

Table 7 Available "free" swell for the different LKAB designs

Design	"Free" Swell
1963	24.0
1983	17.9
2003	5.5

The "confined" swell, on the other hand, is basically scale independent as long as the design powder factor at the toes of the blast holes remains the same, Newman et al (2008) have presented the results of a field test in which a slice of ore was blasted horizontally toward a cave rock filled drift. The "confined" swell values obtained were in the range of 2% to 17%. If one now compares the swell desired by rock materials (magnetite, for example) as shown in Table 6 to that available ("free" plus "confined"), it is clear that the available swell is much less, especially for the very large scale designs. This difference is not reflected in the sand models where the material is fully loosened at the beginning and the swell is uniformly distributed.

3.2.2 Flow described with respect to the "free" swell component

As has been shown, in large scale sublevel caving, the 'free' swell is small and located in the sublevel drift which is easily accessed by only a small part of the overall ring. In this section, a somewhat simplified explanation of the potential flow mechanisms involved will be presented using the geometry of Kiruna as the example. It is patterned after an explanation originally proposed by Larsson (1996) to explain the observed pulsation phenomenon. It will be assumed that the magnetite in the ring has a density of 4.6 t/m^3, would like to swell 50% during blasting (Larsson, 2007). The basic situation is shown in longitudinal section in Figure 15.

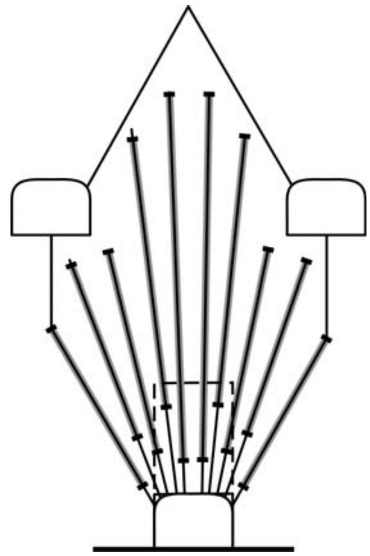

Figure 15 Longitudinal view through the large scale sublevel cave at Kiruna with the available 50% free swell volume limit superimposed.

The cave mechanism will be summarized as a series of steps with corresponding figures:

<u>Step 1.</u> The transverse section taken through the sublevel drift before blasting the next ring is shown in Figure 16. The angle of the front is at the angle of repose of the material which is of the order of 38 to 40^o. An angle of 39^o is assumed. It will be assumed that the brow is intact although, unfortunately, this is often not the case.

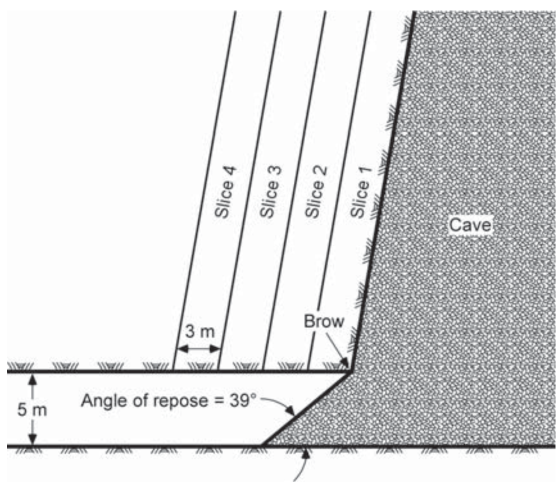

Figure 16 Situation prior to blasting the next ring

<u>Step 2.</u> During blasting it is assumed that it is only the ore directly above the drift is allowed to swell and that swell is assumed to be 50%. The situation after blasting is shown in Figure 17.

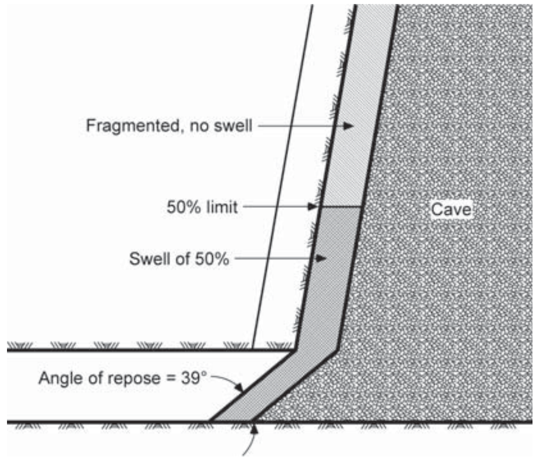

Figure 17 Situation after blasting

The material at the bottom portion of the slice falls down into the drift and fully swells. The front angle is at the angle of repose and material higher in the ring fully swells until all the available swell volume has been taken. The possible swell against the cave is not considered here but will be addressed in the next section. In summary, the lower part of the ring directly over the sublevel drift has first call on the volume presented by the drift and it takes advantage of this opportunity.

Step 3. The load out of the slice now begins. As the loading front is steepened up, nothing happens until a critical angle which is of the order of 45 to 50^o is reached. At this point in time, a volume V_o has been removed. See Figure 18.

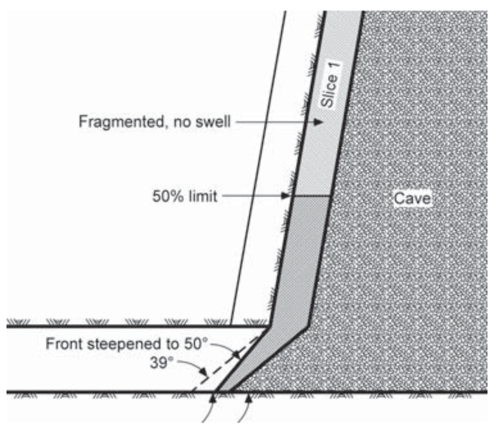

Figure 18 Steepening of the front

At this stage, the front becomes unstable and collapses. Assuming that the cave is stable and the fully swelled ore simply slides downward, the void V_o is transferred to the top of the swelled portion. The material in between has simply slid down without further bulking. See Figure 19.

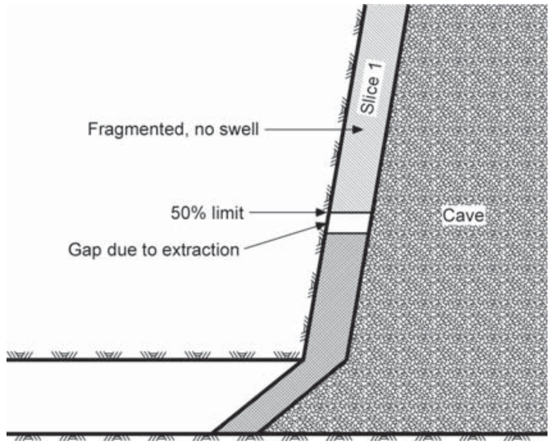

Figure 19 Formation of the void

Step 4. The process of loading front steepening, flowing, sliding, steepening, flowing, sliding etc continues until all of the swelled ore in the slice has been drawn out. The final geometry is shown in Figure 20.

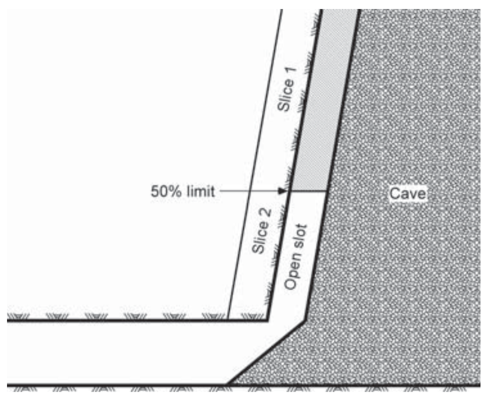

Figure 20 Geometry after the initially swelled ore has been removed

Step 5. If the overlying ore in the slice has not moved due to the confinement of the front-lying cave, a hang-up situation exists. In order for the ore to move, it is necessary to release the confinement on the ore slice provided by the front-lying cave. There are a number of possible remedies including extraction (loading out) of cave material to steepen the cave front, the setting of explosive charges, the use of high pressure water, drilling from the adjacent drifts, etc.

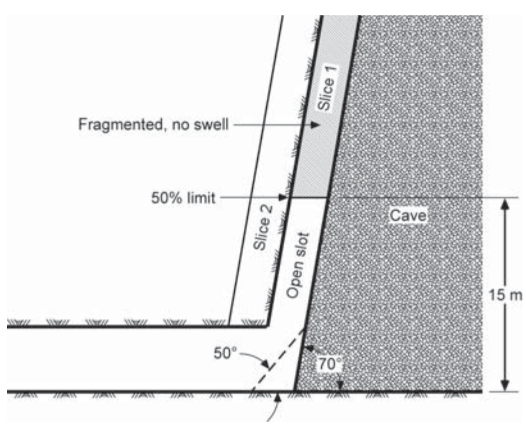

Figure 21 Continued mucking of the cave to induce instability

With luck and given a little time, the cave might become unstable by itself. When the cave moves, it removes the constraint from the ore making up the immediate roof of the slot. With that, the material at the cave front, the overlying ore in the slice, and possibly the ore at the slot sides, compete for the new void volume. The result is an eclectic mixture of materials at the drawpoint front. Figure 22 shows one possibility.

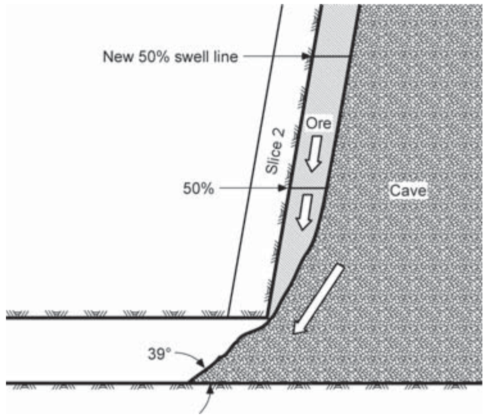

Figure 22 Situation after collapse of the cave front

Step 6. In the case shown, it is assumed that the ore at the sides has remained in place and some of the previously unswelled ore in the slice has swelled to 50%. A new 50% swell line is established with the ore material lying above that line fragmented but without swell. The extraction process continues but now with waste followed by the overlying ore. This is reflected in the pulsation observed in the LKAB marker tests. With extraction, a gap forms under the “New 50% swell line.” One possibility is that the loader operator pulls all of the material and an open slot is again formed. In Sweden, this condition is sometimes referred to as “high hang.” As before, one must do something to get the cave to move and to loosen its grip on the confined ore slice.

A second possibility is that in the drawing process, some high cave material moves into the open slot thereby freeing a portion of the above lying ore and allowing it to swell. A new 50% swell line is thereby established. A logical position for the “high hang” to occur is where the specific charge is the highest. This position is shown in Figure 23.

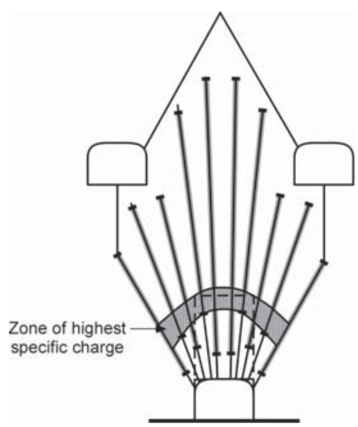

Figure 23 Zone where the specific charge is highest

Step 7. This intermittent flow process is reflected in the ore-waste pulsation observed at the draw point. At times the front can appear to consist of nearly only waste and the loader operator is very inclined to stop loading. However, by continuing the loading process, the ore may reappear. On the other hand, one might have to continue loading cave for a long time before the ore magically appears again. Thus, as indicated by

Quinteiro et al (2001), special loading control procedures are needed to properly follow the extraction process.

Step 8. Ideally all of the ore in the slice eventually moves down to the drawpoint. Towards the top of the ring, the fragmentation is poor due to the low specific charge. With the transfer of the void from the bottom of the flow up through the column, these are blocks have poor mobility. Thus it is easier for the smaller waste/cave blocks to fill the voids. Under these circumstances, the cave in the upper portion of the slice has a higher mobility than that of the ore and fills the slot without the confined ore moving downward. In this case, when the next slice is blasted, the upper portion of the slice does not get well-fragmented so the extraction is poor. Eventually a roof of ore is built up (in Swedish it is designated as "roof remaining") and a new opening slot may be required to re-initiate the process.

Step 9. The flow process described primarily involves the ore located directly over the extraction drift since it is that with the most favorable gravity component. The occurrence of hang-ups, for example, allows the cave to wander somewhat from the largely vertical orientation but not much. After hang-up removal it quickly returns to an upward extension rather than a lateral growth. This is the reason for the relatively narrow cave width noted in the marker tests.

Step 10. The replenishment of the cave rock source from above is interesting in that it often occurs through a vertical column of very limited lateral extent. At Kiruna, for example, the cylinders corresponding to particular draws can sometimes be observed on the surface.

3.2.3 The "confined" swell contribution to flow

A horizontal section taken through a position in a ring is shown in Figure 24.

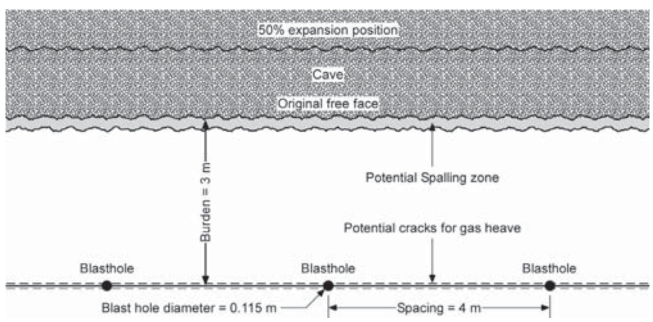

Figure 24 Horizontal section taken through the blasted ring

There are two possible mechanisms for creating swell under these confined conditions. The first is through spalling at the face of the slice (ore-cave interface) due to the explosion generated shock wave. Since the characteristic impedance of the cave is much less than that of the ore, the explosion generated compressive wave impinging on the interface will reflect in tension. The amount of spalling will depend on the amplitude of the shock wave and the tensile strength of the rock. According to Johansson and Persson (1970), spalling in granite will occur when

$$\frac{\text{Charge weight/m of charge}}{(\text{burden})^2} \geq 5 \text{ kg/m}^3 \qquad (1)$$

Assuming a hole diameter of 115mm, the use of ANFO with a density of 0.8 g/cm^3 , and a burden of 3m, one finds that

$$\frac{\text{Charge weight/m of charge}}{(\text{burden})^2} = 0.9 \text{ kg/m}^3 \qquad (2)$$

If the rock involved were granite, no spalling would be expected. If spalls would be produced, the forward movement of these spalls would be restrained by the passive pressure generated by the cave. Hence the potential for swell creation due to spalling is expected to be very small.

The other possibility for swell space generation is through the action of the gas pressure. One could imagine a crack forming between neighboring holes in the ring and the explosive gases flowing into the gap. The gas pressure would then semi-statically force the ring material against the cave. Assuming the explosive to be ANFO with an explosion pressure of 1600 MPa, adiabatic expansion ($\gamma = 3$) of the gas in the borehole, a slot 0.01m wide by 1m high and 4m long (extending between adjacent holes), the pressure in the gap would be about 8 MPa. If the slot were only 0.005m wide, the slot pressure under the same conditions would be 38 MPa. These driving pressures have to be compared to the passive pressure (P_P) on the slice due to the cave (Coates, 1981)

$$P_P = \rho z\left[\tan^2(45 + \varphi/2)\right] \quad (3)$$

Where

P_P = passive pressure (MPa)
ρ = density of the cave (MPa/m)
Z = height of the cave (m)
φ = angle of internal friction (^{o})

Assuming that the mining depth is 500m, the density of the cave material is 1.5 t/m^3 and the angle of internal friction is 30^o, the passive pressure from the cave (that which would oppose the action of the gas pressure) would be

$$P_P = 0.015(500)(3) = 22.5MPa$$

So, depending upon the slot width, there could be some initial movement against the cave but it would be small. The measurements reported by Newman et al (2008) confirm the predicted small magnitude of the spalling/swell movement in the direction of the cave.

Based upon these simple calculations and confirmed by field observations, it appears that the opportunities to increase the "confined" swell contribution appear to be small.

3.4 Effect of scale

There has been an emphasis on increased scale both in the open pits and underground. In the underground environment, drifting is generally much more expensive on a volume basis than is stoping. Hence, one has tried to minimize the specific development or alternatively maximize the tons extracted per meter of development drift. This has been facilitated by equipment manufacturers supplying the drilling machines required for the drilling of long, straight holes. In reviewing the LKAB designs summarized in Table 1, one can see that over the period of time shown the tons per ring have increased by a factor of about 15 and the tons/meter of drift by a factor of nearly 8. With the increase in scale, the area in contact with the front-lying cave has increased by a factor of 8.8 whereas the tonnage per ring has increased by a factor of 16.6. Thus, the waste contact area /ton of ore volume is much improved by increasing the scale and one might expect to be able to recover the ore with less dilution. If the swell-based draw behavior explanation offered above is accepted, then as the scale is increased, the height over which the fight for swell volume between ore and cave will proceed to very great heights. In the upper part of the ore column, the fragmentation will be poor due to the low specific charge (related to the fan drilling geometry), the more mobile cave should be the winner. Therefore, there must be a practical limit to sublevel height where the late stage extraction high dilution and poor recovery overwhelm the development savings associated with the very large-scale. The determination of this limit requires a comprehensive program of mine testing involving markers. Based upon the available marker studies it appears that the sublevel drift spacing should be of the order of 2.5 times the drift width independent of scale.

4 Looking a little back in time and then to the future

4.1 Conventional design concepts, Kvapil (1982, 1992)

Kvapil (1982) has presented design guidance based on the model tests for use with small to medium scale sublevel caving. One begins with the geometry of the extraction ellipsoid for the material and the expected fragmentation. Curves based on a large number of sand-based model experiments are provided in this regard. The size and shape of the sublevel drift is largely determined by equipment requirements. Given the size of the expected fragmentation and the density of the ore, one can estimate the maximum expected width (W_T), depth (D_T) and height (H_T) of the extraction ellipsoid. The mid-height of the sublevel drift (H_s) should be placed where the extraction ellipse is the widest. This occurs at about

$$H_s = 2/3\ H_T \qquad (4)$$

An approximate horizontal spacing of the horizontal drift axes (S_D) can be determined knowing W_T. The following rules apply:

For $h_s \leq 18$m

$$S_D < \frac{W_T}{0.6} \qquad (5)$$

For $h_s > 18$m

$$S_D < \frac{W_T}{0.65} \qquad (6)$$

In conventional sublevel caving,

$$S_D \leq H_s \qquad (7)$$

The thickness of the blasted slice (the burden, B) is given by

$$B \leq \frac{D_T}{2} \qquad (8)$$

Where

$$D_T \leq W_T/2 \qquad (9)$$

As noted, based upon improved longhole drilling precision in recent years, there has been a tendency to increase the sublevel height. Kvapil (1992) has presented some ideas regarding how this change would modify these design rules and cave performance.

4.2 Layout suggestions, Bullock and Hustrulid (2001)

Bullock and Hustrulid (2001) presented some practical rules for making initial sublevel caving designs. These rules are summarized below:

- sublevel drift size (width (W_D) and height (H_D)): defined based on equipment operating space requirements
- sublevel interval (H_S): based on the ability to drill long, straight holes. This, in turn, is based on the hole diameter (D)
- hole diameter (D): based on the available drilling equipment and the ability to charge long holes
- spacing of the sublevel drifts (S_D): based on a construction procedure involving the known sublevel drift geometry and 70^o side draw angles
- ring spacing (burden (B)):

$$B = 20\ D, \text{ for ANFO} \qquad (10)$$

$$B = 25\ D, \text{ more energetic explosives on a bulk strength basis} \qquad (11)$$

- hole toe spacing (S_T): based upon the burden

$$S_T = 1.3\ B \qquad (12)$$

- front inclination: 70 to 80 degrees (forward)

If it is assumed that

D = 115mm

Drift dimensions: 7m wide by 5m high

Explosive: Emulsion (high bulk energy)

Sublevel interval: 25m based on drilling ability

One finds that the remaining dimensions are:

Sublevel drift spacing: 22m

Burden: 3m

Toe spacing: 4m

Front inclination: 80^{o} selected

Assuming a side draw angle of 70^{o} one obtains the values for sublevel drift spacing as a function of sublevel height for two sublevel drift geometries in Table 8.

Table 8 Sublevel height and sublevel drift spacing for two different drift sizes using Bullock and Hustrulid (2001)

Drift size: 5m by 5m	
H_S (m)	S_D (m)
15	12
20	16
25	20
30	23
Drift size: 7m by 5m	
H_S (m)	S_D (m)
15	14
20	18
25	22
30	25

A reasonable extraction ellipse extending over two sublevels and between adjacent sublevel drift centerlines is obtained.

4.3 New layout rules based upon marker test input

Based upon the results of the three marker tests, it appears that the elliptical shape of the draw body does not apply. Rather the draw width (D_W) is a constant times the width of the sublevel drift (W_D). Based upon the very limited information available, one obtains

$$D_W = 1.4\ W_D \qquad (13)$$

This means that the center-to-center spacing of the sublevel drifts should be

$$S_D = 2.4\ W_D \qquad (14)$$

Table 9 Sublevel height and sublevel drift spacing for two different drift sizes using equation (14)

Drift size: 5m by 5m	
H_S (m)	S_D (m)
15	12
20	12
25	12
30	12
Drift size: 7m by 5m	
H_S (m)	S_D (m)
15	17
20	17
25	17
30	17

In comparing the results in Table 9 to those given in Table 8, one finds that they are in general agreement for the lowest sublevel heights for the two drift geometries but then they diverge significantly.

Based upon the new information, it is suggested that the Bullock and Hustrulid (2001) design rules be modified. These modified rules are summarized below:

- sublevel drift size (width (W_D) and height (H_D)): determined based on equipment
- sublevel interval (H_S): the theoretical maximum value is based on the ability to drill long, straight holes. This, in turn, is based on the hole diameter (D). The actual limit is based on recovery and dilution considerations which are due to managing ore/waste pulsation.
- hole diameter (D): based on the available drilling equipment and the ability to charge long holes
- spacing of the sublevel drifts (S_D):

$$S_D = 2.4\ W_D \tag{14}$$

- ring spacing (burden (B)): based upon the damage radius (R_d) concept discussed by Hustrulid and Johnson (2008)

$$B = 2\ R_d \tag{15}$$

Where

$$R_d\ /\ r_h = 20\sqrt{\frac{P_{e\,Exp}}{P_{e\,ANFO}}}\ \sqrt{\frac{2.65}{\rho_{rock}}} \tag{16}$$

R_d = damage radius (m)
r_h = hole radius (m)
$P_{e\,Exp}$ = explosion pressure for the explosive
$P_{e\,ANFO}$ = explosion pressure for ANFO = 1600 MPa
ρ_{rock} = rock density (g/cm^3)
2.65 = density of typical rock (g/cm^3)

- hole toe spacing (S_T): based upon the burden

$$S_T = 1.3\ B$$

- spacing for parallel holes (SP): based upon the burden

$$S_P = B \tag{17}$$

- front inclination: 70 - 80 degrees (forward)

If it is assumed that

D = 115mm

Drift dimensions: 7m wide by 5m high

Explosive: Emulsion ($P_{e\,Exp}$ = 3900 MPa)

Rock density = 4.6 g/cm^3

Sublevel interval: 25m based on drilling ability and control of pulsation

One finds that the remaining dimensions are:

Sublevel drift spacing: 17m

Burden: 2.7m

Toe spacing (fanned): 3.5m

Toe spacing (parallel): 3m

Front inclination: 80^{o} selected

It is noted that the new sublevel drift spacing rule has very limited basis and must be carefully complemented with further testing. There also needs to be practical limits due to the pulsation effect.

4.4 Implications for future sublevel caving designs

The results of the marker studies would indicate some major changes are required in current sublevel caving designs. Assuming that the drift width is not changed, it would indicate that the sublevel drift spacing should be reduced and thus the overall mining scale would decrease (presuming no change in the sublevel height). One way of maintaining the current scale is to increase the width of the sublevel drift. Figure 25 shows one possibility.

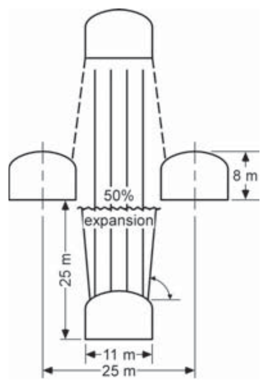

Figure 25 Silo design with super-scale extraction drifts, patterned after Kvapil (1992)

This has advantages with respect to the silo shape and the parallel hole drilling. However, one must be concerned with geomechanics issues (drift and brow stability). Furthermore, the draw must be well controlled over the entire face.

If one wants to preserve the specific development ratios in place today, one would need to increase the sublevel height. However, this has problems with hole deviation, maintenance of long holes, charging of very long holes, and dealing with ore/waste pulsation over a much longer draw duration. This seems like a very difficult alternative to achieve on a day to day basis. On this basis, it would seem that in the future mining companies will be looking toward smaller scale designs than today and not larger. The current very large-scale designs may actually be too large-scale.

4.5 Front caving implications

The paper has only dealt with standard sublevel caving. There are a number of variants, however. Front caving is a variety of the sublevel caving technique which is quite often used It is, for example, a very interesting technique for the creation of the undercut required in block and panel caving. However, it is very important that the undercut be completely formed. The marker studies would indicate that the flow stream is much narrower than previously thought. If rock mass flow does not occur over the full drilled width, the remaining portions could form remnants and transmit loads to the production level with catastrophic consequences. This means that current undercut designs based upon front caving will have to be re-evaluated.

4.6 Future possibilities to maintain/increase scale

There are two possibilities, at least, to try and maintain or possibly even grow the scales used today. One possibility deals with using more of the sublevel drift for swell than just that taken by the ore falling down. This involves changing the blasting pattern and initiation sequence so that the ore at the lower part of the ring is propelled far out into the drift. A second possibility which also involves a change in the blasting is to use the available swell space more effectively. This means only permitting the ore in the lower part of the ring to only swell 20% rather than 50%. This would thereby increase by a factor of 2.5 the amount of ore in the ring which has a chance to swell. Accomplishing both of these possibilities should be well within the capabilities of electronic detonators with very precise timing.

A problem with today's typical ring drilling design is that the hole spacing changes from very small near the drift to large at the hole ends. The parallel hole design used in the silo design avoids this problem. Without a major change in drift width, one is confined to a rather narrow pattern. Figure 26 shows one possible futuristic design involving special drilling technology and the blasting innovations which better use the available "free" swell space.

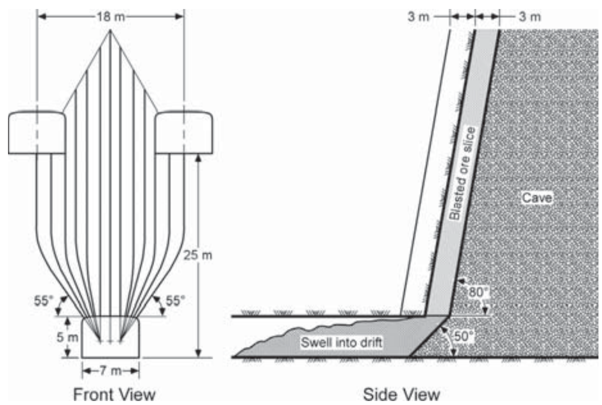

Figure 26 New possibilities for large-scale sublevel caving

The design presents an opportunity to achieve improved fragmentation, an increase in ore mobility, and a more uniform distribution of ore mobility over a much wider front. An understanding of how the ore actually flows in sublevel caving will lead to better designs. The marker studies are an important step along that path.

4.7 Future studies

In closing, the authors believe that it is time to seriously revisit the recommendation made by Janelid (1961) nearly 50 years ago with regard to small-scale sublevel caving

"The results achieved have been so encouraging that continued research work is well justified since the economic benefits which can be achieved through the development of the correct method are extraordinarily large."

One important result of this recommendation in 1971 was the pursuance of a marker study at the Grängesberg mine and a complementary modeling program. In retrospect, a closer evaluation of the results collected at the time might have had a substantial impact on the designs made and executed since then. For example, in spite of the close spacing of the drifts and the care taken in retreating adjacent faces, no apparent interactive draw was observed. This was one of the conclusions of the Perseverance studies conducted in 2004.

In spite of their obvious value, field studies are few and far between in the mining business. In addition, if conducted, it is very difficult for others to access the results and perhaps gain and offer new insights. This must change if the mining business is to meet the technical, economic and safety challenges the future has to offer.

There is a real danger that today's sublevel caving designs are far from optimum due to a poor understanding of the fundamental processes involved. In the past, the application of sublevel caving has primarily been to iron ore, particularly magnetite, which because of its very forgiving magnetic property, permits easy and inexpensive separation from the waste. The same is not true with other minerals, for example copper porphyry and gold ores. For these, it is very expensive to separate ore and waste. It would appear that prior to fully committing to any sublevel caving design, a pilot project should be run with a carefully planned and executed program of data collection. One very important piece of information to be extracted is the draw width. It is also very important to develop the required draw control techniques to be applied in the mine. As was pointed out, with ore/waste pulsation which is inherent in very high draw designs, practical draw control becomes very difficult. Visual viewing of the cave front is not enough.

Thank you for the time spent in reading our contribution.

Acknowledgements

The authors would like to express their appreciation to Carlos Quinteiro, Håkan Selldén, Jan-Olov Nilsson, Anders Nordqvist and Lars Larsson (retired) of LKAB who have provided some of the background information used in this paper.

Disclaimer

Much of the material in this paper references LKAB's Kiruna mine since the senior author had the privilege of working there for several years. However, the ideas and opinions expressed are those of the authors alone and do not necessarily reflect the views of LKAB.

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