The Control of a Brine Inflow and Support to the Resulting Solution Cavern in a Conventional Saskatchewan Potash Mine

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In 1985, Agrium's potash mining operations, then known as Cominco Fertilizers Inc., experienced a roof failure followed quickly by a brine inflow at the end of an abandoned mining panel. Over a decade was spent rehabilitating openings into the area, carrying out exploration drilling to determine the source and brine flow patterns, and finally to successfully control and grout the inflow. Drilling outlined a number of discrete flow paths in the overlying limestone which, when relieved through cased drill holes, stopped the uncontrolled flow into the mine and allowed grouting around the inflow site using cement and flyash mixtures, ultrafine grouts and chemical grouts to take place in a "no flow" condition. Although the location of the inflow was known, it was several years before access was actually gained, through difficult ground conditions, to allow monitoring the effectiveness of grouting operations. During this period a large solution cavern over 30m in diameter and 20m in height had developed. Approximately 4m of the overlying Red Bed shales and 3m of limestone had collapsed into the cavern. The cavern had to be backfilled to a workable room height and the roof safely supported to eliminate further deterioration and caving. An overview of the inflow and grouting methods is discussed however emphasis is on the creative support techniques developed to cope with the severely limited access to the cavern.

1. INTRODUCTION

Agrium is one of North America's largest fertilizer companies and a major producer of nitrogen-based fertilizers, potash and phosphate. Based in Calgary, Alberta, Canada, the corporation has production facilities in Alberta, Ontario and Saskatchewan, Canada, Nebraska and Texas in the United States, and Argentina.

Mining takes place in the upper part of the Patience Lake member of the Prairie Evaporite Formation (Devonian) at a depth of 1100m. The Prairie Evaporite is underlain by the dolomitic limestone of the Winnipegosis Formation. Above the salt lies the 4m thick 2nd Red Beds, shale unit, followed by the Dawson Bay limestone (Mackintosh 1995,p.383).

Mining takes place using borers to cut 1600m long rooms 5.5m wide by 3.35m high. On completion of this 1st pass, the borer is turned around and a second pass cut widening the room to a final width of 10.2m. A series of five such rooms separated by 6.7m to 7.6m pillars makes up a panel. Panels are separated by 45m support pillars (Mackintosh 1977, p.3).

In May of 1985, a roof failure in one of the abandoned turn-around areas in the extreme West end of the mine was followed quickly by the smell of hydrogen sulfide gas. On investigation, a small brine flow was found running down an adjacent ventilation drift identifying the source as being in panel 17. Several months of entry rehabilitation and new entry cutting were required to access the area, create appropriate sumps, and install pumping systems to remove the stored brine to surface. By December of 1986 the first exploration holes were being drilled from underground around the inflow site.

2. DRILL HOLE CONFIGURATION

Drill holes consist of a 4.5m long by 100mm diameter first casing cemented into the salt. This is followed by an NW second casing installed to a point 2.4m to 3m below the anticipated contact between the Prairie Evaporite salt and the 2nd Red Beds. A third and final BW casing is cemented 2.4m to 3m above the contact between the 2nd Red Beds and the Dawson Bay limestone. The final hole penetrated the limestone until brine flow was encountered or to an elevation 28m to 30m above the mine level. All drilling was carried out through a high pressure valve and blow out preventer.

3.0 INFLOW DESCRIPTION

By early 1988 sixteen drill holes had been drilled around the inflow area, and it was confirmed as coming from the turn-around area for room 2 of panel 17. Flow rates and pressure data from these holes indicated that the inflow brine source was in the limestone coming from the West, past the edge of mine workings. Several holes intersected flow rates in excess of 1000 litres per minute, much greater than the 300 to 400 litres per minute coming into the mine indicating a restriction between the inflow site and the brine source and the potential for much greater flow than was being experienced.

By the end of 1991 over 100 holes had been drilled and had defined a zone of brine filled fractures along the West boundary of the mine from 250m South of the room containing the inflow to 300m North. Of this 550m, a region approximately 350m wide contained fractures with flows greater than 225 litres per minute.

3.1 Initial Grouting

The initial grouting plan was to grout from the North and South regions along the West boundary of the mine, narrowing the zone of brine filled fractures until a full seal along the West was achieved. As grouting progressed and pressures increased, longer relief holes would reduce the flow and pressure to allow good grout penetration. After achieving a seal along the West boundary, the fractures leading from the West to the inflow site would be grouted.

A considerable amount of time and a tremendous number of holes were drilled to try and seal the North and South limits of the brine filled zone. As many holes were grouted in actively flowing water, it was impossible to determine the effectiveness and quality of the grouting.

At several times during the life of the project, grouting took a hiatus while entries into the area were rehabilitated. As the inflow area was in an old mined out area, continuous closure of openings presented a constant need for scaling, bolting and re-cutting to maintain suitable access. These activities made up at least a third of the time spent on the inflow project and it became obvious that the plan needed to be revisited to see what could be done to enhance the progress.

3.2 Revised Grouting Plan

In mid 1994, Mr. Stephen H. E. Phillips of Phillips Mining, Geotechnical and Grouting was retained as primary grouting consultant. A review of the problem by Mr. Phillips resulted in a revised plan which essentially reversed the order and direction of grouting to starting at the inflow site where the problem was small and could be surrounded and working out to the West, progressively filling all pathways. Once the west boundary had been reached, all pathways between the brine source and the inflow site should already have been sealed.

Flow tests carried out on a number of the holes indicated discrete pathways trending in a NE - SW direction. There was good communication between holes along this trend but very poor communication in the NW - SE direction.

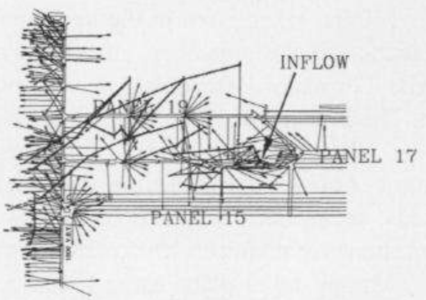

Figure 1. Inflow, drill holes and pathways.

Flowing a hole on one pathway would significantly reduce the pressure and flow on holes on the same path but had little or no effect on holes on adjacent parallel pathways. Also, when the flowing hole was again shut in, the pressures on connected holes took some time to recover indicating it was possible to draw down the area.

The revised plan required additional holes around the inflow site, additional holes between the site and the West boundary to ensure all pathways had been intersected, and additional relief holes along the West boundary to try and stop the uncontrolled flow into the mine so grouting could take place in a no-flow situation. In all, over 400 holes have been drilled to deal with the inflow. An essential component of the plan was to gain access to the inflow site. It was critical that the inflow be monitored to determine if sufficient relief holes were available to stop the flow into the mine and to monitor the effectiveness of grouting.

4. INFLOW SITE

An Alpine AM-50 was used to re-cut panel 17 room 2 from the West towards the turn-around area. As cutting approached the turn-around, a sink hole or vertical chimney feature was encountered where brine had dissolved a semi-circular channel in the overlying salt. This chimney was approximately 1.2m by 1.8m and appeared to go straight up to the overlying limestone. A .9m diameter culvert with internal rungs was installed in order to gain access to the top of the salt. It was expected that the collapsed salt covered by 2nd Red Bed rubble would be found which could be cleaned up to provide a workable floor to access any cracks which could be monitored and possibly chinked should they leak grout. Instead, the culvert emerged about halfway up a huge cavern, 33.5m across and almost 20m in height. The collapsed salt had been completely dissolved and the bottom of the cavern filled with the 2nd Red Bed rubble and blocks of limestone. The back formed a smooth arch with the highest point approximately 3m into the limestone. The inflow brine could be seen running from numerous joints in the limestone.

5. GROUTING

With access now available, the effect of flowing relief holes could be monitored. Holes that effected the inflow were allowed to flow. This initial flow was in the order of 2,000 litres per minute dropping over a period of about two weeks to 585 litres per minute and eventually to 500 litres per minute. This stopped any inflow into the cavern and dried up many holes between the West boundary and the inflow. When grouting was initiated around the cavern, cement was immediately seen coming from the joints that had previously produced brine. On continued grouting and thickening the mix, these leaks stopped and the holes were pressured to 2.75 MPa. The low pressure was to avoid any hydro-fracturing around the cavern.

Grouting progressed from East to West using type 30 cements and flyash and ultrafine cements depending upon the acceptance. Once grouted, holes were re-drilled and re-grouted as required, again using ultrafine and type 30 cements. Grouting pressures progressively increase to the west to just over 17 MPa. When all holes would no longer accept ultrafine cements, they were chemically grouted with Avanti AV-100. Again holes were re-drilled and after they would no longer accept any grout they were brushed clean of any chemical, flushed clean and pumped full of a thick mix of type 30 cement. Once this was set the valves were removed.

5.1 Cavern Support and Grouting

To be sure the inflow problem has been successfully sealed it will be necessary to monitor the cavern for several years. It was decide to backfill the cavern to within 3m - 3.5m of the back and install support packs. A saltcrete made up of screened run of mine muck from entry rehabilitation mixed with cement and flyash was though to be the easiest fill to produce. Because of the limited access up the .9m culvert, any fill material must be pumpable and flow readily with almost an infinite slump. M.D. Haug and Associates of Saskatoon, Saskatchewan, Canada, carried out a series of tests on the salt/potash and cement mixture to determine the slump and unconfined strength of various water/cement ratios, percent cementitious binder, and percent of flyash replacement in the binder. Cement contents of 10, 15 and 20 percent, of which the flyash replacement was 0, 30 and 50 percent were tested. The final mix determined to suit the backfill needs contained 15% - 20% cement of which 50% was flyash, and sufficient brine to provide a 1.2 brine/cement ratio.

Three HW holes were drilled from the mine level up into the cavern. A casing shoe allowed drilling to continue through the loose rubble and the casing could be pushed up close to the back. Filling took place through the casings using a concrete pump and mixing station set up on the mining level. Pumping distance was approximately 100m. Two 250mm dia. HDPE pipes were also drilled into the cavern as exhaust ports for air coming up the culvert. Backfilling started by pumping up a cement slurry without any aggregate in an attempt at filling void spaces and consolidating the rubble.

Figure 2. Filling from casing.

When cement could be seen above the base of the rubble, the -20mm mesh screened aggregate was introduced. Mixes with lower cementitious content providing strengths of between 4 and 5.5 MPa were used for most of the filling. The final 3.5m was filled with material containing 20% cement having strengths of 6.5 to 7 MPa to provide a stronger floor beam for supporting packs. By moving between holes it was possible to achieve a relatively flat floor. Twice the access culvert had to be extended in 2.4m increments. As the fill approached the intended height, a hose could be attached to the casing and the saltcrete distributed around the cavern to help level the floor.

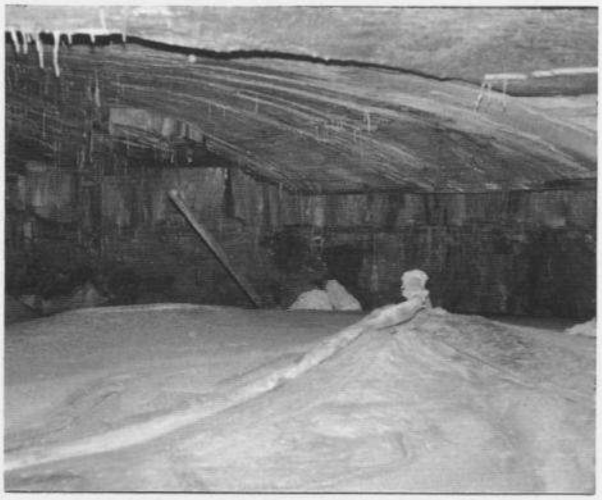

Figure 3. Fill and distribution hose.

Once the backfill was to the desired height, saltcrete pillars or cribs were poured to provide support to the back.

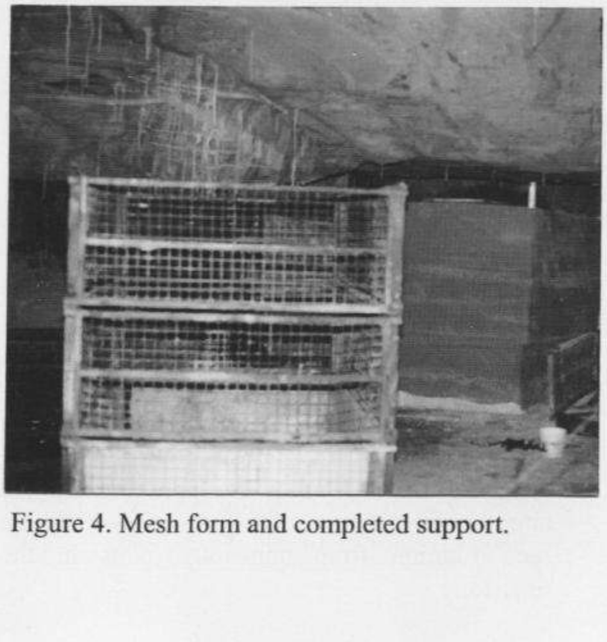

Figure 4. Mesh form and completed support.

Forms constructed of steel tubing and wire mesh in .6m by 2.4m and .6m by 1.5m rectangles were hoisted up the culvert manway and pinned together to form a 1.5m by 2.4m rectangular form, 1.8m to 2.4m high.

A fabric bag was sewn to fit into this form and saltcrete was again pumped through the casings and hose to fill these bags.

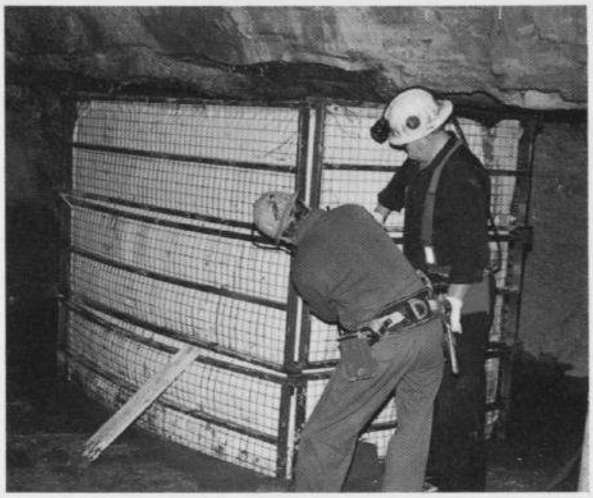

Figure 5. Completed form and fabric liner.

The forms would be half filled then allowed to set before continuing to final height to avoid stressing the forms or breaking the bags.

Custom fabricated geotextile bags designed to accept cement were then placed on top of the pillar and pumped full of thick cement slurry.

Figure 6. Completed support.

The filled bags conformed to the contours of the back. It should be pointed out that the ground temperature is 30.5^{o} C. While these supports were being placed the curing cement raised the temperature to over 43^{o} C resulting in a difficult working environment.

With the back thus supported a number of holes were drilled from inside the cavern using a bar mounted diamond drill. Several of these holes intersected fractures connected with holes drilled around the cavern from the mining level. Grouting through these cavern holes connected with the surrounding holes and joints in the cavern. These joints were chinked with wedges, the cement allowed to set, then the holes were re-drilled and re-grouted ensuring all pathways into the cavern were sealed.

6. CONCLUSION

At the time of writing, drill holes were being final plugged with cement after being chemically grouted. It is planned to close the valves on the relief holes and allow the area to re-charge. Pressures on the relief holes will be monitored and a micro-seismic monitoring system has been installed around the area to detect any new inflows in otherwise inaccessible mined out areas. The cavern will be monitored for leaks and, if it remains dry for several years, will be totally backfilled and the relief holes grouted and abandoned.

I wish to thank Agrium for allowing this paper to be presented, the drilling and grouting crews who persevered through difficult conditions, particularily when filling and supporting the cavern where the only access was through the .9m dia. manway, and to Mr. Stephen Phillips for his guidance throughout the grouting period.

REFERENCES

1. A. D. Mackintosh, 7th Williston Basin Symposium (1995) p383.
2. A. D. Mackintosh, 6th International Strata Control Conference (1977) p3.

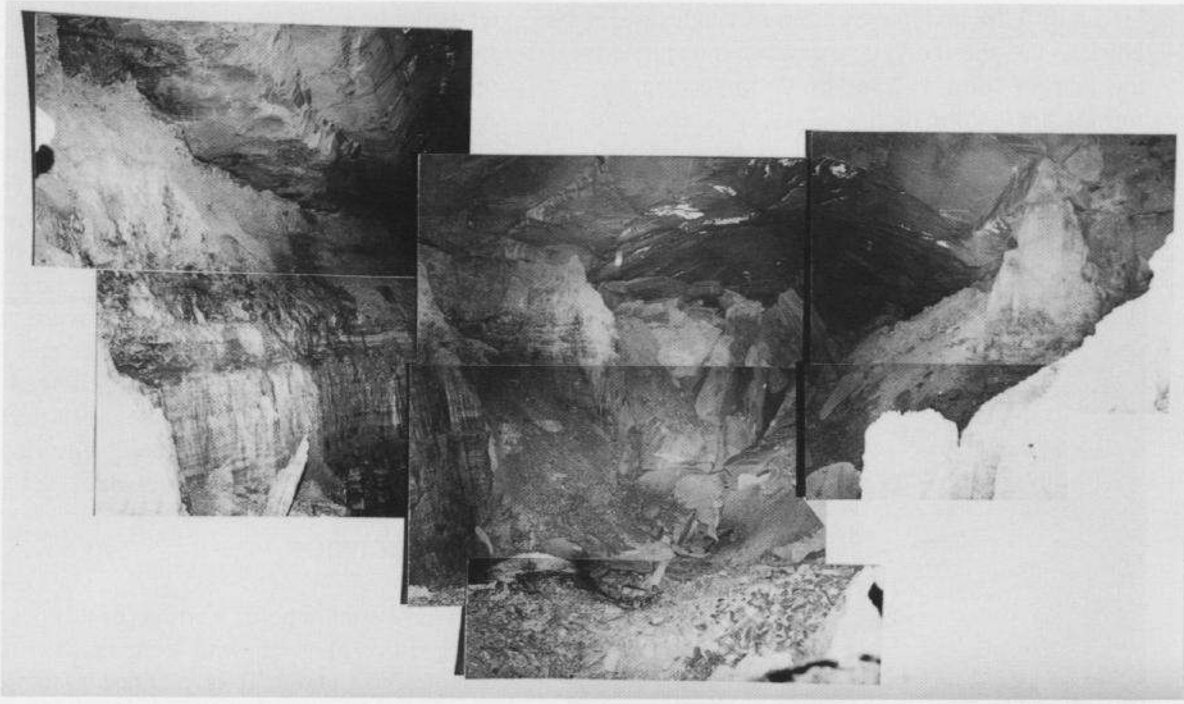

Figure 7. Cavern as seen from the manway prior to filling.

Figure 8. Cavern and manway taken from figure 7.