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HYDRAULIC SLUICED LONGWALL MINING WITHOUT SUPPORTS

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ABSTRACT: A concept for a mining method developed for a series of thin soft sloping seams between competent roof and floor strata is presented. The method proposed has the form of a retreating longwall where the mining method is by hydraulic sluicing. The face is essentially formed by drilling a directionally controlled borehole between gateroads. The directional drilling assembly is then removed from the drill-string and replaced by a jetting bit which is used to sluice the coal from between the borehole and the goaf. The process may then be repeated by drilling another hole parallel to the first and the process of mining operation is repeated.

BACKGROUND

Present longwall practice

Longwall mining is a capital intensive inflexible system which can yield very high production when things go right. It is generally suitable for operation in geologically uniform situations where the panel width can be maintained and where the panel length is adequate to justify the cost of moving the machinery between panels. It is limited in the range of height in which it can be used to those that permit miners to work safely. This means about 1.5 to 4.0 m, or higher if top level caving is used. Longwall mining has been used on slopes to about 25^{o} but at these angles there is a constant battle to prevent the equipment walking down hill with the consequence that production is diminished.

Hydraulic mining

Hydraulic mining has been in existence for many years. One of the most notable examples of its use was Mitsui Sunagawa mine in Hokkaido, Japan in an approximately 6 m thick weak (3 MPa uniaxial compressive strength) seam at a dip of approximately 70^{o}. There the coal was sluiced by the use of a monitor via adits driven in rock, which intersected the seam to form monitor bays and draw points. The water used for this operation was delivered to the 22 mm diameter monitor nozzle at pressures of 10 to 15 MPa and a flow rate of 3 m^3/min, (Gray, 1980). This corresponds to a nozzle velocity of 132 m/s.

This was a gassy mine was a gassy mine which had a very strict regime of gas drainage to avoid outbursts.

Following the closure of Sunagawa mine the pumps and monitor were transferred to the Strongman mine in New Zealand (Duncan, 1998). The use of the monitor was to be from a series of sub horizontal roadways driven in the sloping coal seam. A similar plan was prepared for the underground part of the Burton Mine in North Queensland. However this was never put into operation. The ill fated Pike River mine also planned to use hydraulic extraction (Whittall, 2006).

The fundamental limitation of all the hydraulic mining systems is the amount of roadway that must be driven to the volume of coal that is extracted. The reason for this is the limited effective range, approximately 30 m, of the water jet produced by the monitor. This limitation is brought about by dispersal of the jet and by limited visibility for the operator.

Directional drilling

Over the past 25 years, directionally controlled drilling has been used in underground mining so that holes of 600 m length can readily be drilled in seam. These can be extended to about 1500 m with more effort and a drop in drilling rate (Gray, 1994). The drilling system that has been almost universally adopted for this purpose comprises one that uses a poly crystalline cutter drag bit rotated by a down-hole motor. This bottom hole assembly has as part of it a bent sub which can be orientated so that the drill may drill in a preferred direction chosen by the operator. This orientation is achieved by rotating the drill string

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to a chosen tool face position under the real time guidance of an electronic survey system, which contains triaxial magnetometers and accelerometers. Thus it is possible to drill around rolls in the seam or to re-join the seam if it is disrupted by a fault. The angular build rate that may be achieved by such a system commonly lies in the range of 0.5 to 1.0^o per metre. Rotation of the drill string may be used to make the bottom hole assembly drill a relatively straight hole.

The drill pipe used for this process has, in the Australian context, been almost universally NQ or more recently NRQ type manufactured by Boart Longyear. This high tensile steel pipe (620 MPa yield strength) with heat treated tool joints has an outside diameter of 69.9 mm (2 3/4") and an inside diameter of 60.3 mm (2 3/8"). It has been generally used to drill boreholes using a 96 mm drill bit. This drill pipe has been found to be particularly useful because it only weighs 23 kg per 3 m length and is readily handled by underground drill operators.

Water jet drilling

Over many years and several research projects water jet drilling in coals has been shown to be feasible. Some of the most important work in this area was conducted by Kennerley (1990) and others that followed at the University of Queensland culminating in the tight radius drilling system that permits drilling at the end of a hydraulic hose. The drilling pressures used have varied between about 50 and 120 MPa with flow rates of approximately 0.2 m^3/min. Also it has been shown that it is possible to cut slots with the water jet using lateral jetting nozzles at the drill bit.

There is a significant difference in the flow rates and pressures used between water jet drilling and hydraulic sluice mining. One is a cutting process which uses high pressure water while the other is more dependent on mass flow and momentum to deliver a force to the coal while pressurising cleats to assist the disintegration of the coal. The former would appear to rely more on the effects of cavitation to cut the coal.

THE PROPOSED MINING METHOD

The concept

The mining method proposed uses a combination of the technologies described above. It works as a form of longwall between gate roads which are driven on the across dip with sufficient gradient to allow flow coal and water and have a ditch formed in them to carry the slurry produced by mining. The difference is that the face does not have the conventional powered hydraulic supports nor a shearer and armoured conveyor. Rather the face is formed by drilling from one gateroad across the panel to the next gateroad. This drilling would be accomplished by using directional drilling techniques to cover a panel that might be between 100 and 500 m in width. Once the drill has reached the far gateroad the directional drilling assembly is removed and replaced by a lateral jetting assembly. Water is then pumped through the drill string and out of the lateral jet to erode the coal which flows under gravity back to the lower gateroad. The drill string is withdrawn during this process as the coal is eroded. Once the string is fully withdrawn it is moved with the drill down dip in the gateroad and the process is repeated. It is envisaged that adjacent boreholes would be placed 5 to 25 m apart, depending on local conditions.
The erosion is accomplished either completely between the boreholes or leaving some form of narrow pillar of material so as to control roof failure.

Drilling may be accomplished from the upper gateroad to the lower one with erosion taking place on withdrawal of the drill string uphill. In this case the slurry formed by mining would flow down through what could potentially be goaf. By angling the face line by choice of drilling direction a face may be maintained so that the slurry will flow down the intersection of the retreating face and the floor. Alternatively a narrow pillar can be left so that the slurry flows down an enlarged, eroded hole that does not collapse immediately.

Alternatively the drilling may be undertaken from the lower gateroad to the upper one to ease the return of cuttings. In this case the hole would have to be of reasonably large diameter (0.3 m) and may need to be enlarged by reaming to this size. Such reaming could be undertaken by either the use of water jets or a mechanical rotary reamer. The jetting process for mining would then be undertaken. This would involve jetting from the top down thus eroding coal that flows back down the enlarged borehole.
Once the slurry has reached the lower gateroad it can be in the ditch or flume to a sump for pumping to surface as slurry or for local separation into wetter and drier components which can then be transported

to surface. The version of the operation that uses drilling from the upper gate road is shown Figures 1 and 2.

Figure 1 shows a panel in a view that is perpendicular to the seam being mined. The coal to be mined is to the right between the face and the main headings. The goaf is to the left of the face. The face lies between the upper (top of drawing) and lower gateroads. Drilling takes place from a drill site of the lower of the upper gateroads. Drilling is accomplished to the upper of the lower gateroads. The directional drilling assembly is replaced by a lateral jetting nozzle and mining is accomplished by jetting to form jet cut mining face (3-3) and flow is down the advancing face edge to the gateroad where it flows in a ditch to the sump at the bottom right hand corner of the figure. The previously mined panel lies to the top of the figure. An un-eroded pillar of coal is left below the upper gateroad to protect the drill and crew.

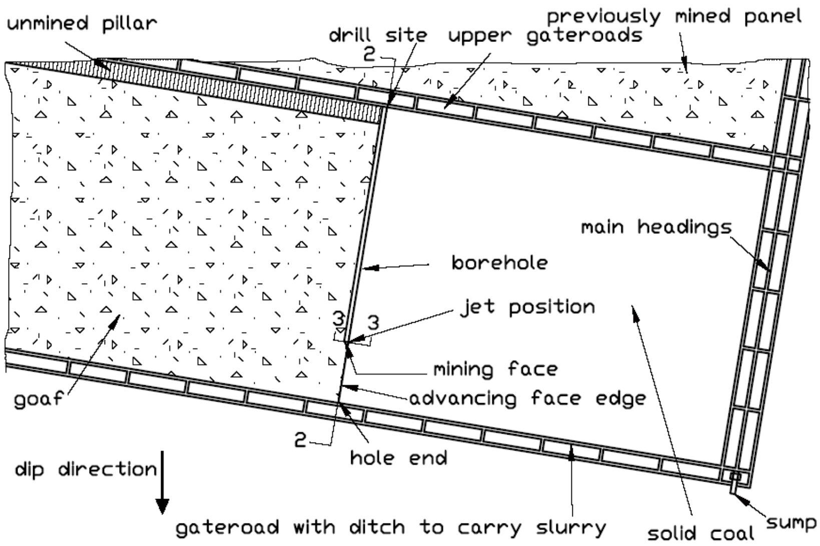

Figure 1 - View of hydraulic mining longwall panel perpendicular to the seam

Figure 2 shows an almost complete section (2-2) along the face of Figure 1. The goaf of the previously mined panel is shown in the upper right hand corner above the upper of the upper gateroads. The drill is in the lower of the upper gateroads. It has been used to drill to the upper of the lower gateroads where it will be fitted with a lateral jetting bit. The drill string has then been withdrawn while jetting the section marked as the eroded zone. The slurry produced from this has flowed to the ditch in the lower gateroad for transport to the sump.

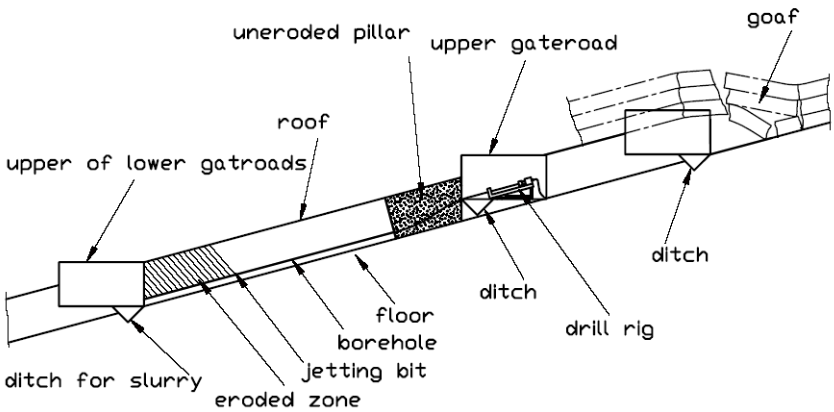

Figure 2 - Section of hydraulic mining system roughly along section 2-2 from Figure 1

Figure 3 shows section 3-3 from Figure 1. It shows the jetting bit which is protruding from the borehole with a lateral jet which cuts a mining face roughly perpendicular to the advancing face. A goaf is shown being formed to the left of the drawing. The slurry runs down the intersection of the floor and advancing face (out of the page) toward the lower gate road.

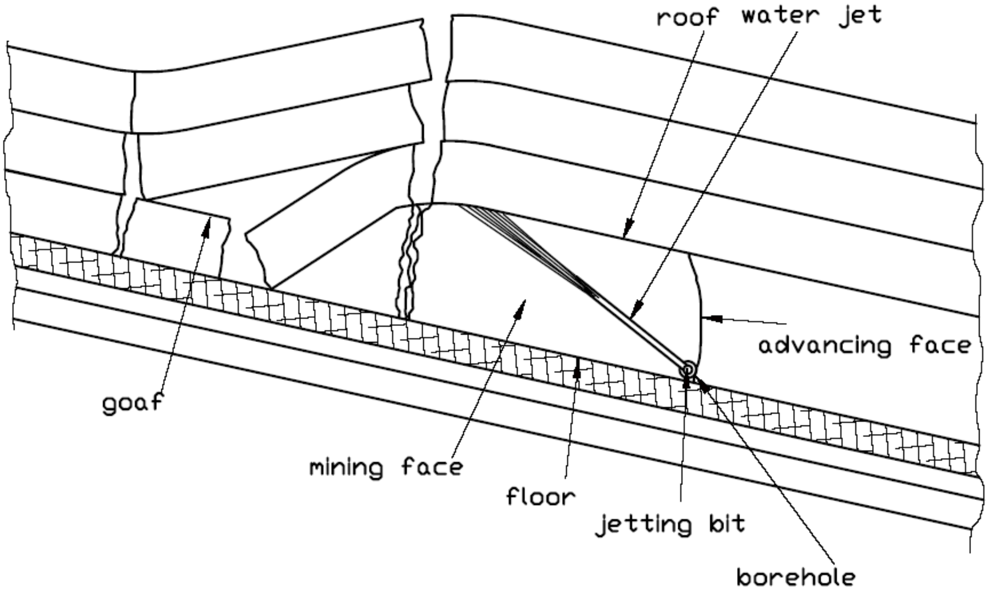

Figure 3 - View up dip of hydraulic mining showing mining face: follows section 3-3 from Figure 1

Drilling

Rather than drill with the standard NQ or NRQ drill pipe currently used in directional drilling operations it is proposed to move to a large diameter pipe. This size increase is to allow the pipe to have a larger fluid flow rate capacity, so that it will be more robust to withstand greater torque to turn it. Such a pipe is readily sourced by simply increasing by one size to HRQ drill pipe also manufactured by Boart Longyear. The outside diameter of this is 88.9 mm (3 ½") with an inside diameter of 77.8 mm. This is comparatively light at 35 kg per 3 m length or could be reduced to 23 kg per 2 m length. The drill hole would be drilled using a 3 ½" down-hole motor to about 150 mm. The survey system would be similar to that used currently for directional drilling. Heavier drill pipe could be used for greater robustness but the manual handling of such pipe would be a burden. Standard oilfield tool joints will normally be externally upset. Such upset tool joints would pose a problem for extraction if the hole collapsed. Flush external, internally upset oilfield type tool joints would lead to excessive pressure loss at each tool joint for the flows being considered.

Sluicing hydraulics

The HRQ drill pipe proposed has been tested and found satisfactory to 35 MPa pressure using some specialist tool joint seals. The volumetric capacity over a length of 300 m of such drill pipe is 2.9 m^3/min for a 2.5 MPa pressure drop within the pipe. Such a pressure drop is quite acceptable and delivers a flow very similar to that used at Sunagawa. To deliver 12 MPa at the nozzle will require a total of about 14.5 MPa to be available at the top of the drill string. This pressure and volumetric flow can be delivered by larger oilfield triplex mud pumps that are available [13.8 MPa (2000 psi), 750 USGPM] or possibly more readily from pairs of smaller mud pumps. Such flows could be readily reticulated around a mine using 200 mm pipes with little pressure loss.

The nozzle could possibly be better designed than those used at Sunagawa which had a very high pressure loss coefficient. There is a desire to manufacture the jetting bit in a slick form so that it does not become trapped in the event of hole collapse. The jet proposed would be formed out of a gradual taper and bend housed within the same diameter section as the drill pipe. This can be achieved using spark erosion manufacturing techniques in very hard material to resist wear.

The flow of the slurry along the advancing face of the goaf edge or the hole which is being eroded will be one of the controlling factors in the success of the mining method. The flow here will depend primarily on the slope available for the fluid to run down and whether the goaf collapses right up to the retreating face edge. In the latter case the process of leaving an un-eroded pillar of coal between the current mining zone and the goaf may be adopted to temporarily hold the area open. The prime requirement is to be able to break the coal into sufficiently small pieces that it is carried down slope in a turbulent flow. Experience would suggest that the slope must be higher than 5^o for this to occur. Slopes closer to 10^o would provide better security against a blockage.

The movement of the slurry in the ditches of the gateroads is essentially a problem of channel flow. Fortunately coal has a comparatively low density and is therefore more easily transported in water than denser rock which may need periodic manual removal. If required the use of a simple lining of the ditch using sheets of polyethylene will reduce surface roughness and erosion thus speeding flow and reducing maintenance requirements.

Strata control

One of the keys to the successful implementation of the system is that of strata control. The roof must stay up for a while before the goaf forms so that flow from the mining face can take place and the floor not be excessively eroded. The general requirement is therefore for a soft easily erodible coal seam between relatively strong roof and floor. This is a requirement that is met in some geological settings.

Ventilation

The need to ventilate the face poses significant limitations on the system. If the goaf collapses right up to the face then the ventilation along the face will be lost. It is therefore necessary to be able to arrange the ventilation so that it can be maintained along the top and bottom gate roads if this situation occurs. This will permit the drilling and starting of another face adjacent to that is being lost. The question of how the water jet mining face is regarded from a ventilation viewpoint needs to be considered. Is it a face that must be kept ventilated at all times, or is it a part of the goaf where there will be always at some location an explosive mixture of gas and air? The latter means that the mining method is simpler to use. The former would require the formation of the goaf proper to be delayed by leaving narrow pillars behind that maintain an opening for ventilation. The actual face where the water jet would be applied would not however be ventilated. This is not an issue if it is regarded as a wet borehole with no risk of ignition, nor if the coal is not gassy.

Jet control

The water jet control technology need not be initially in excess of anything that is used for current directional drilling. All that is required is an indication of radial position, or tool face angle in drilling terms, of the water jet. The operator can then play this jet in the required direction for a suitable period, or until coal ceases to be produced, and then move the drill string longitudinally to commence the sluicing of the next section of coal. If a hard roof and floor exist then they will limit the extent of erosion. Refinements could be made by the introduction of improved monitoring systems to determine whether the coal had been sluiced away. Such systems might use visual imaging, acoustic imaging or variants involving listening to transmitted sound of the jet hitting the side of the opening. These are not however considered to be necessary for the system to be trialled.

CONCLUSIONS

The system proposed is a simple low cost method that is designed to permit the establishment of a form of continuous panel mining, which will achieve significant extraction ratios with respectable production for an order of magnitude or less cost than a conventional longwall. It has the requirement that the seam to be mined slopes adequately for hydraulic mining practises to be used. However it has the advantage that it can be used around faults as there is no restriction that requires the panel width to remain constant. Neither is there a concern with a fault transecting a panel. The panel will simply be drilled and sluiced up to the fault and then recommenced on its other side. There is no need to move a major face line incorporating powered hydraulic supports, conveyor and shearer. It is also suited to seams that are too narrow to be mined by conventional longwalls.

The possibility exists that the face may collapse but this is not a disaster as it can be readily re-established by drilling a new hole. Even the loss of all the mining equipment that exists along the face occurred it is unlikely to cost more than $0.5 million. Compare this with the cost of an iron bound longwall!

The system is simple and could be applied using existing technology now. All that is needed is a seam of greater than 10^o slope and preferably with a moderately strong roof and floor. For a trial the seam would preferably not be gassy but the potential exists within the system to re-enter gas drainage holes drilled within the seam and use them as the basis for the mining method.

Patents have been applied for to cover the methods described.

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