Rapid Excavation and Tunneling Techniques

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Introduction

The topic of Rapid Excavation in Tunneling[1] provides an informative illustration of how research related to one aspect of subsurface engineering developed in the US from the mid-1960's -70's –and some developments since that time.

The Cold War era had produced a concern, especially in the aftermath of the Cuban Missile Crisis of 1962,[2] that senior members of the US Administration be safe, allowing government to continue to function effectively in the event of a catastrophic nuclear attack on Washington DC. Establishing an underground command center deep under the Capital was considered to be the most viable option.[3] Speed of excavation of tunnels was the primary concern. Government individuals informally expressed the need for a dramatic *("order of magnitude or more"[4])* increase in tunneling excavation rates. [The 1998 report "Use of Underground Facilities to Protect Critical Infrastructures" https://www.nap.edu/catalog/6285/use-of-underground-facilities-to-protect-critical-infrastructures-summary-of indicates the continuing interest in this attribute of the subsurface, even though the Cold War had ended.]

The National Academy of Engineering (NAE) was founded in 1964 –some two years after the Cuban Missile Crisis. One of the groups established by NAE was the Committee on Rapid Excavation Techniques (NAS/NAE/NRC).[5]

In 1971, the National Science Foundation was being urged by the Nixon Administration to establish research programs intended to address national priorities.[6] The RANN (Research Applied to National Needs (1971-78)) was established. 'Rapid Excavation' was again one of the research priorities.

The topic was timely; the Robbins Company had pioneered the development of Tunnel Boring Machines (TBM's) in the US[7]; the Washington Metro project had started recently (1969); the prospect of the Channel Tunnel between France and the UK was under active consideration; the *First* North American *Rapid Excavation and Tunneling Conference* was held in *Chicago 1972*'[8]; the International Tunnelling (and Underground Space) Association (ITA) was founded in 1974 [the term "Underground Space" was added later]. The Martin Marrietta Company was developing a drill to obtain rock cores from the Moon[9], as part of the Apollo 13 mission. Launched on April 11, 1970, this capsule did not land on the Moon https://en.wikipedia.org/wiki/Apollo_13

University and industrial research groups participated in the RANN Rapid Excavation program. Various techniques of weakening the rock were considered;- chemical attack; electrical shock; high pressure water jet; the "Nuclear Subterrene"[10;] - and other concepts were all examined. The book, *Novel Drilling Techniques*, Pergamon Press, Oxford, England, 1968, by W.C. Maurer, and the report *Rock Fragmentation* (1976)

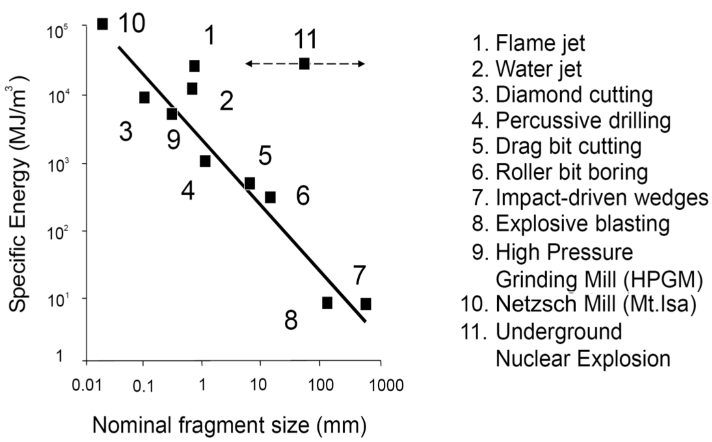


***Figure 1**. Specific Energy as a Function of Nominal Fragment Size for Various Rock Breaking Methods in Quartzite [modified from the original by Cook and Joughin. (1970)].*

http://www.osti.gov/scitech/servlets/purl/5342136 provide good descriptions of the techniques, discussions and conclusions at that time. The report concluded that high pressure water jets had the highest potential for successful application to rock excavation. The Discussion and Conclusions section of the Rock Fragmentation report notes (p.7)

"The schemes which combined water-jets with mechanical breakage devices appeared to receive the most favorable review. The energy of water-jets has the capacity to produce deep kerfs, without a significant removal of rock mass. A secondary system can then take advantage of the free surface created by the kerf."

The Mining Research Laboratory of the Chamber of Mines, in South Africa, faced with the problem of excavating the hard abrasive and gold-bearing Witwatersrand quartzite, also considered a variety of alternative rock fragmentation techniques. **Figure 1**, a result from these studies, illustrates that the energy required to crush a unit volume of rock increases rapidly as the size of the crushed rock decreases. This is also a major issue in the field of comminution. [See 'Energy Requirements' in https://en.wikipedia.org/wiki/Comminution].

Although total energy requirements are a consideration in tunnel excavation, e.g. much of the energy consumed in fragmenting rock re-appears as heat – a serious concern in deep (hot and humid) mines – for most practical applications the overall 'effectiveness' of the system is the prime concern. In the case of tunneling this is essentially the maximum rate of advance. In recent years, developments in Tunnel Boring Machines (TBM) have made this system the preferred option, especially for long tunnels where rock conditions are reasonably uniform.

TBM Excavation

TBM excavation technology has developed considerably over the past several decades and is applied widely (**Figure 2a**), although there are a variety of efforts to develop alternative techniques of continuous excavation by machine

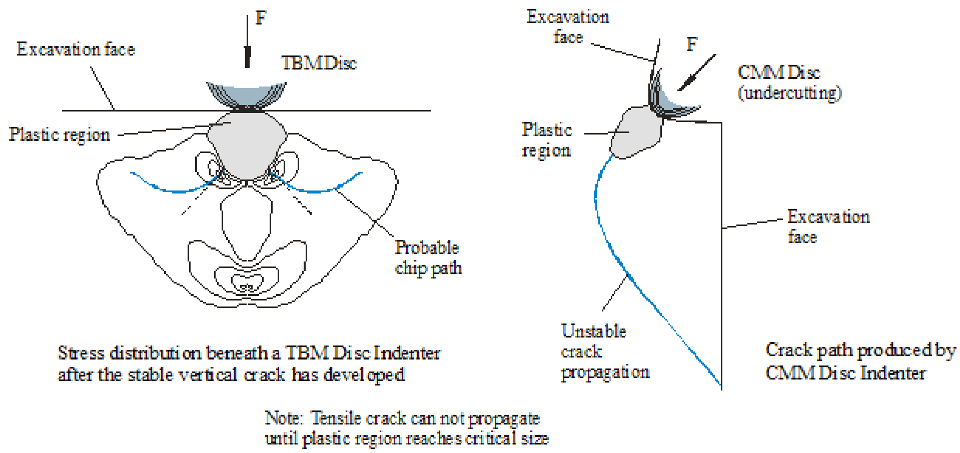
(a) *(b)*

Figure 2. *(a) Robbins TBM (Tunnel Boring Machine) for the 43.5-km long 10-m diameter water supply tunnel – AMR Project Andhra Pradesh, India[11]. (b) Wirth CMM (Continuous Mining Machine.)[12]*

The WIRTH CMM (Continuous Mining Machine) **Figure 2b** 2b used the 'undercutter' principle of rock fragmentation. This machine had four roller discs, one mounted on each of four rotating arms to cut the rock. The roller disc on Arm 1 cut a saucer-shaped depression into the tunnel face [shown in red in **Figure 2b**]. Discs 2, 3 and 4, spaced 120° apart on the central arbor, cut outwards from this depression towards the tunnel periphery. The trajectory of these arms was computer controlled, to allow a non circular tunnel to be excavated, including 'essentially' flat -floor or rectangular profiles (with rounded corners). The arms are retractable so that the face of the excavation is readily accessible when necessary. Unlike the standard TBM, a substantial component of the reaction forces of the cutters is directed normal to the tunnel axis -and 'sustained' by the arbor. This substantially reduces the intensity of the forces that must be sustained by the 'Grippers' that are pressed hydraulically against the tunnel wall in a standard TBM.

It is suggested that this 'undercutting' system has the advantage that the rock fragments are broken in tension whereas the standard TBM causes failure in compression, hence requiring higher forces and higher energy. Although true in part, the undercutting process is more complex.

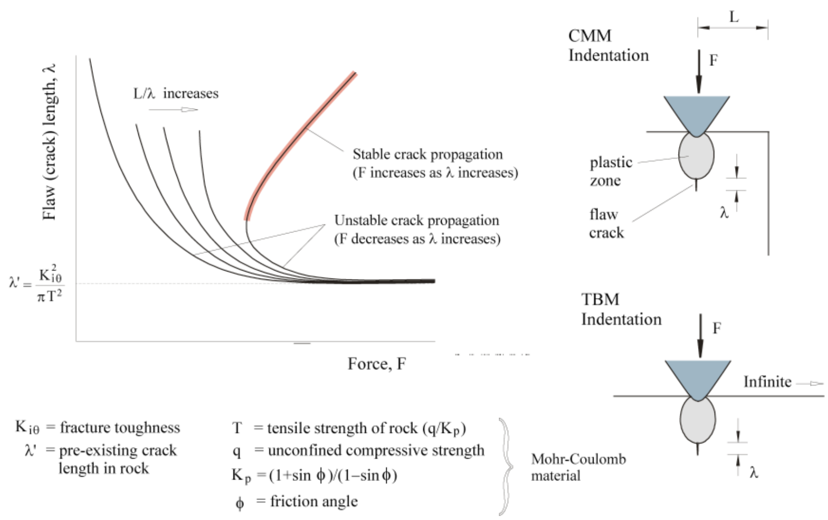
As illustrated in **Figure 3**, the TBM disc applies a force normal to the rock that is increased until the rock fractures. The applied force creates a zone of intense multi-axial compression ('plastic region' in **Figure 3**) immediately below the disc contact. A tensile stress develops directly beneath the plastic zone, initiating a tensile crack. This crack lengthens as the force F is increased, extending directly along the axis of the applied force i.e. into the (semi-infinite) interior of the rock mass. This crack is 'stable' i.e. requiring the force to increase continually in order to extend. The plastic region also induces tension, albeit of lower intensity, in the rock to the sides of the axis of loading. Eventually, this 'off-axis' tension will reach the tensile strength of the rock and initiate off-axis cracks [shown in blue in the 'TBM diagram' (**Figure 3**, upper left). These cracks can then propagate towards the free surface (Excavation face in **Figure 3**, upper left) to generate a rock chip.



Disc tip radius	CMM Indenter Force (N/mm of contact length)			
	S = 25 mm	S = 50 mm	S = 100 mm	S = ∞
6 mm	1,700 (74%)	2,310 (100%)	2,700 (117%)	3,000 (130%)
8 mm		2,500 (100%)	3,150 (126%)	(3,500?) (140%?)
12 mm		3,150 (100%)	3,600 (114%)	4,200 (133%)

Note: Indenter Force Increase is small for large increase in cutting depth S.

Figure 3. *Comparison of chip formation in classical TBM and in CMM (undercutting).*



Relationship between applied force on an indenter and the length of edge crack (λ) beneath the plastic crushed zone

Figure 4. *Relationship between applied force on an indenter and the length of edge crack (λ) beneath the plastic crushed zone.*

With the CMM, the disc forces are directed asymmetrically with respect to the excavation face. Initially, the tensile force develops beneath the plastic zone, much as in the case of the TBM disc loading. As the CMM load increases, the initial tensile crack 'senses' the excavation face and can propagate towards it. The path of the CMM crack is shown in blue in the upper right hand diagram of **Figure 3**.

Figure 4 shows a plot of the force F that must be applied to a CMM cutter disc [force F applied *parallel* to the tunnel face] and to a standard TBM roller disc [F applied *normal* to the tunnel face] in order to extend the crack (flaw) length (λ), once the force reaches the minimum value required to start a pre-existing crack (length λ') in the rock to extend from the plastic zone. As seen in the upper left diagram of **Figure 4**, once the flaw λ' starts to extend, the force F required to extend the crack *decreases* - the crack is *unstable* and will propagate spontaneously. In the case of the TBM, once the critical flaw starts to extend, it continues spontaneously over a short increase in length, but then further extension is *stable* –i.e. the force must be increased continuously in order for the crack to extend. (This region is shown in red in the graph in **Figure 4**). It should be noted that the analysis leading to the graph in **Figure 4** does not consider the onset of the lateral (off-axis) cracks in **Figure 3** (upper left diagram).

The CMM indenter force required to cause the crack to extend for various disc tip radii for the situation shown in **Figure 4** (upper right) and distance from the free surface [L in **Figure 4** (but shown as S in the table of **Figure 3**)] is tabulated in **Figure 3**.

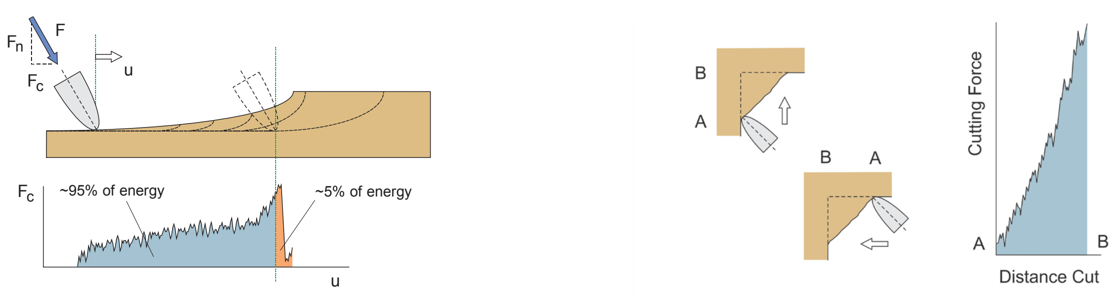
The table in **Figure 3** reveals a very important feature of the CMM – the cutting force does not increase in direct proportion to the depth of cut. Thus, F increases by 60% (2,700/1,700) for an increase in depth of 400% (100 mm/25mm).

Field experiments with the CMM carried out in Ruhr Sandstone confirmed that the CMM does indeed produce large fragments (**Figure 5**)[13]. The fine rock cuttings visible towards the top of the pile in **Figure 5** are attributable, in large part, to "geometrical effects" that may be overlooked in theoretical analyses. In **Figures 4** and **5**, for example, the analysis assumes that the disc cutter acts on a rectangular ledge of rock. Once this ledge is broken away as part of the large fragment, the disc must then remove the rock from the curved lower surface created by the detachment of the large fragment –as a precursor to re-developing the steep face ahead of the disc– needed to detach another large fragment [**Figure 6a**]. Study of the disc cutting forces as the disc traverses this precursor path show that the energy consumed in the process is of the order of 95% of the total –with the large fragments accounting for no more than ~5%!

It is essential that, as the tunnel advances, the contour dimensions (or 'gage') be maintained. For drill and blast systems, there will be small local variations (of the order of cm) around the mean dimensions of the tunnel, but boring machines maintain tighter tolerances.

Figure 5. *Rock cutting produced by CMM in Ruhr Sandstone.*

The energy consumed in this process of 'cutting gage' is the same whether the cutter advances parallel or normal to the tunnel axis (**Figure 6b**) i.e. be it TBM, CMM or any other machine system. Gage cutting will produce very fine particles ('dust')–and hence will result in high cutting forces. These high forces will be maintained across the entire tunnel periphery, where the tools are traveling at peak speeds. This combination indicates that cutting tool wear will be most severe during the gage cutting process. This suggests that use of high pressure water jets as an aid to mechanical cutters for gage cutting may be worthy of serious study. Water does not 'wear' and could help suppress much of the dust at its source.



(a) Linear cutting *(b) Corner cutting*

Figure 6. *Energy dissipated during rock cutting (Schematic).*

Although the Wirth machine has several attractive features for tunneling, it appears currently unable to match the advance rates of the standard TBM.

The undercutting principle used in the Wirth system has gained favor for application to continuous mining machines. The Sandvik reef borer MN220[14] (early prototype known as ARM1100[15]) has been developed for use in South African platinum mines, and the MX650[16] is under construction specifically for rapid mine development. DynaCut[17], the mobile hard rock miner by Joy Global incorporates the Oscillating Disc Cutter (ODC) technology, developed by CRCMining in the early 1990s. ODC applied oscillatory motion on the disc cutter during the cutting process, and it is claimed that this reduces significantly the thrust required for a given advance rate. Dehkhoda and Detournay (2016) in investigation of activated undercutting discs (which they refer to as 'Actuated Disc Cutting') indicate that if the actuation mechanisms (similar to oscillation in ODC) do not reduce the total energy required for cutting, they do indeed reduce the required thrust. Laboratory tests are now underway to establish whether the actuation action of the cutter introduces a rock weakening effect, not considered in Dehkhoda and Detournay (2016). The link https://confluence.csiro.au/display/HRCT/Current+projects discusses these tests and includes a short movie of the cutting action.

As illustrated by the recent papers by Hassani F. et al.(2016) and Toifl, M.et al.(2015) on microwave irradiation of rock, efforts continue to find methods to reduce rock strength and aid rock fragmentation in comminution and underground excavation but, so far, these are largely confined to laboratory studies and have not yet achieved practical application in tunneling.

Considerable research has also been conducted since the RANN program of the 1970's on the development of water jets to aid in rock excavation. High pressure (200-250MPa) pumps and jets are now available, but as yet, to the writers' knowledge, no commercial water jet 'kerf excavation' system has been developed. A review of developments in pulsed water jets for use in rock fragmentation is provided in Dehkhoda (2014). Lui, S. et al. (2016) also describe use of water jets to assist pick cutters fragmentation capabilities, but again these are still in the experimental stage.

In summary, increasing the advance rate of tunneling machines through techniques to improve the efficiency of rock fragmentation has proven to be a formidable challenge

Drill and Blast Excavation

Excavation of tunnels by detonation of explosives in holes drilled in the tunnel face was the only method used prior to introduction of TBM's in the 1960's. The high capital cost of TBM's tends to limit their application to long tunnels for which a circular cross-section is acceptable. Drill and Blast excavation has the drawback that it is a cyclical operation, and requires that personnel be evacuated from the tunnel face during blasting. Also, where excavation rate is concerned, TBM's are usually capable of outperforming current drill and blast operations.

It is interesting to speculate as to the potential for increasing advance rates using the drill and blast principle. **Figure 7** shows, schematically, a typical pattern of blast holes used to fragment the rock to advance the tunnel. The depth of the holes, all drilled essentially parallel to the axis of the tunnel advance, is usually comparable to the width of the tunnel.

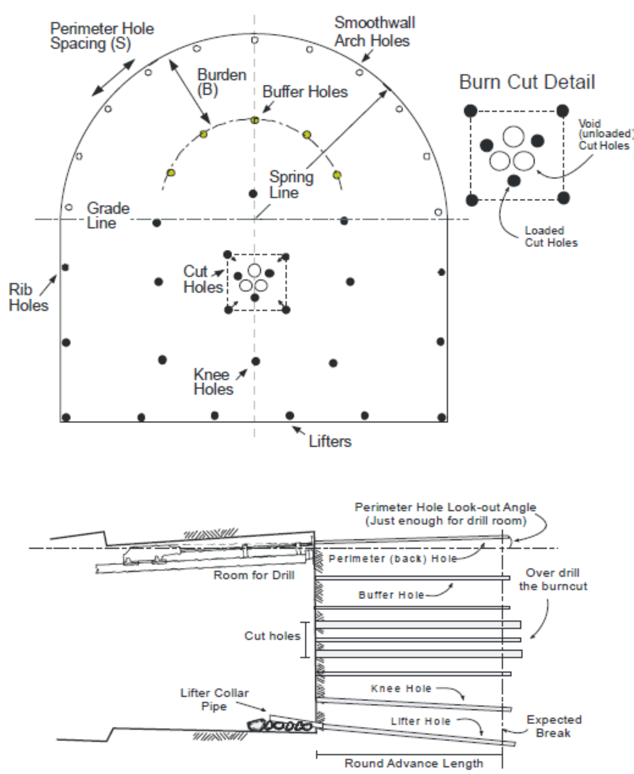
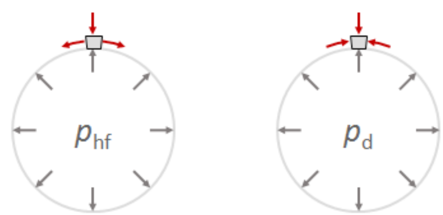


***Figure 7**. Typical Pattern of Drill Holes for Drill & Blast Excavation of a Tunnel – using a 'Burn Cut' [Diagram- Courtesy of Revey Associates].[18]*

In the specific design illustrated in **Figure 7**, a central 'cut' [the dashed square denoted 'Cut Holes'] is developed [see 'Burn Cut Detail"] by detonating the three charged holes immediately around three (larger diameter) uncharged central holes, followed by detonation of the four holes at the corners of the rectangle. The purpose of the cut blast is to create a rectangular void to the full depth (or 'burden') of the holes. Removal of this volume of rock provides a space to accommodate the volumetric expansion of the rock surrounding the cut when it is subsequently detonated – sequentially in stages (at delay intervals of the order of 10 milliseconds each) to the final profile.

The advance per blast (i.e. 'burden') is seen to be approximately equal to the width of the tunnel. Although this and similar blast designs have been developed empirically (i.e. by trial and error) the depth of burden can be explained by considering the in situ stress distribution in the vicinity of a tunnel face. To a rough approximation, the region comprising the tunnel face and sides within one tunnel 'radius' 'a' from the face can be considered as a hemisphere. The stress change ahead of this hemisphere will fall off as $(a/r)^3$ where 'a' is the tunnel 'radius' and 'r' is the radial distance from the center of the tunnel face into the rock. Thus, at the far extent of the drill holes, i.e. approximately r = 2a, the change in stress from the 'pre-tunnel' stress state will be of the order of $(1/2)^3$ or ~12%. At that depth the blast fragments will be constrained from expanding axially towards the face. The fragments are still able to expand radially towards the central cavity, which increases as the rock is detonated in stages (of the order of 10 milliseconds apart) progressively outwards from the central cavity.

The process of rock fracturing by explosive loading differs considerably from that during quasi-static pressurization of a borehole –as occurs, for example, during hydraulic fracturing. This is illustrated in **Figure 8**. In hydraulic fracturing, the fluid pressure in the hole is increased slowly over a period of minutes. The stress increase at the wall of the borehole is 'communicated' to the interior of the rock at the velocity of sound (c) for that rock (c is typically several thousands of meters/second). Thus, the rate of pressure increase occurs very much slower than the change of stress in the rock. The rock expands, and a tensile stress is developed tangential to the applied radial pressure –until the tensile strength of the rock is reached and a single rock fracture is generated and extended radially into the rock mass. (If the hole is in a rock mass that is subject to in situ stresses, then the fracture will develop at the point(s) on the borehole where the compressive stress concentration is at a minimum (i.e. at two, diametrically opposite points on the hole periphery).



(a) Hydraulic Fracturing *(b) Explosive Loading*

(i) Hydraulic Fracturing; Pressurization rate; 0 - 50 MPa[19] in 10 minutes (ii) High Explosives; 0 - 10 GPA in 1μs (iii) Mining Explosives; 0 - 3 GPA in 10μs

Figure 8. *Loading of a borehole by (a) slow and (b) explosive pressurization of a borehole.*

The situation under explosive loading is very different. For high explosives (e.g. TNT), the pressure rises to a very high level (~10,000MPa) in 1 microsecond. At a sonic velocity of, say 5,000 m/s (i.e. 5mm/microsec) the pressure 5mm from the periphery of the hole is still unaffected by the intense pressure at the hole. The dynamic loading is essentially similar to that of pressurizing the very small annulus of loaded rock against a rigid outer boundary. Both the radial and tangential stresses are compressive – and the rock is subjected to intense compression, in all directions radially, tangentially[20] and axially – and is crushed intensively in tri-axial compression. Eventually, tensile stresses and fractures are developed around the outer boundary of the plug region –but gases generated by chemical reactions in the explosive column do not enter the fractures, at least until late in the fragmentation stage.

(a) (b)

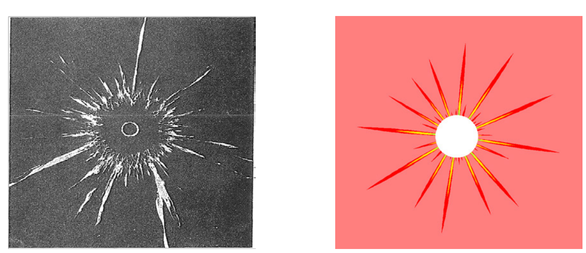


Figure 9. *Fracture development around an explosively loaded borehole.*
(a) in a Plexiglas block[22] (b) numerical simulation

Figure 9a shows a block of Plexiglas, after explosive loading of the central hole. The central region in which there are no discrete fractures is clearly evident. **Figure 9b** shows a numerical simulation of fracture development outside of the pulverized zone. This is intended to illustrate the mechanics of multiple radial fracture generation during explosive loading. When a fracture is generated dynamically, the tangential stress across the fracture surface is reduced rapidly to zero. An 'unloading stress wave' starts to propagate around the hole, at sonic velocity, on each side of the fracture. This unloading wave will counteract (i.e. reduce) the tangential tension generated by the initial explosive loading.

In the ideal case, it may be imagined that a multitude of fractures starts to propagate simultaneously from the hole. If one extends slightly ahead of its neighbors the associated unloading wave will tend to inhibit the growth of these neighbors. Thus, a few fractures only will propagate – the number will depend on the rate at which the shock pressure develops compared to the sonic (unloading) velocity. **Figure 9b**, shows the result of a numerical simulation of fracture generation by explosive loading of a central hole in a block of rock. The block is unloaded at its external boundary. A related phenomenon of (thermally induced) tensile crack propagation in glass has been discussed by Geyer and Nemat-Nasser (1982). If the external boundary is subject to stress - as is the case for a borehole in an underground location, then the external stress field will also act to inhibit extension of cracks normal to the maximum (compressive) in situ stress. This situation is illustrated schematically in **Figure 10**.

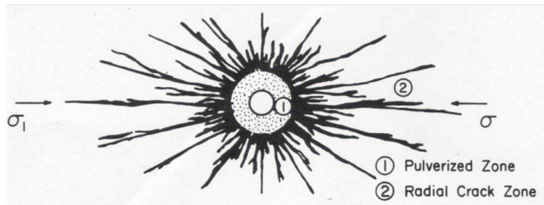


Figure 10. *Influence of in-situ stresses on dynamic fracture development around a blast hole.*[22]

Blo-Up [23] has been used extensively in simulation of blast phenomena, primarily in open pit blasting –but could provide considerable insight on the potential for innovation in drill and blast excavation of tunnels. This is discussed further below.

A 1994 report published by the National Academies Press *Drilling and Excavation Technologies for the Future* (https://www.nap.edu/catalog/2349/drilling-and-excavation-technologies-for-the-future) presents an excellent outline of opportunities to improve these technologies and recommendations for federally funded research to achieve these advances. As noted in the Executive Summary

> *"Long-term payoffs will come from advances in basic R&D to assemble these individual components into a* ***smart drilling system****—a system capable of sensing and adapting to conditions around and ahead of the drill bit."*

Regrettably, no Federal action was taken on the recommendations of this important report. Several major developments have resulted from research carried out by petroleum companies and service companies in the intervening two decades. The 'smart system(s)' of directional drilling has been the key to development of the massive non-conventional oil and gas resources across North America. Similar deposits exist also across the globe.

The improved ability to model the processes of rock fragmentation by explosives, as described above, suggests perhaps another possibility for taking an important step forward in the quest to achieve substantial increase in the rate of tunnel advance.

Returning to the discussion of the drill and blast cycle using the burn cut and a series of parallel drill holes, it is not unusual, today, to replace the cut holes of **Figure 7** with a borehole large enough to create the central cavity directly without blasting. This would provide a void volume into which the surrounding rock could be blasted. Although the lack of the tunnel face would be a constraint on expansion, it is not clear how much of a limitation this would impose (e.g. a somewhat larger hole may offset this shortcoming). Detonation of explosives in horizontal holes extending several kilometers has been used as a technique to generate a system of fractures to stimulate oil and gas recovery from non-conventional formations. Is it conceivable that somewhat analogous technologies might allow the rock even kilometers ahead of a proposed tunnel alignment to be fragmented and weakened sufficiently to allow much more rapid excavation than currently envisaged? The numerical modeling tools available today allow such a possibility to be investigated**.**

Acknowledgments

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Endnotes

[1] https://books.google.com/books/about/The_National_Academy_of_Engineering.html?id=TjwrAAAAYAAJ (See p.138)

[2] https://en.wikipedia.org/wiki/Talk:Cuban_Missile_Crisis

[3] http://coldwar-c4i.net/DUCC/presmem1.htm.

[4] Personal communication.

[5] Prof E. P. Pfleider (Univ. of Minnesota), Chairman.

[6] https://nsf.gov/about/history/nsf50/nsf8816.jsp See Chapter IV *Tumultuous Times*; para. 6 et seq.

[7] James Robbins had developed a successful TBM in 1952 for the Oahu Dam project, and was the world leader in TBM technology.[See also https://en.wikipedia.org/wiki/Tunnel_boring_machine]

[8] It has been held biennially since 1971 and typically attracts over 1000 participants.

[9] Gravity on the Moon is 1/6 of that on Earth – but the rock strength is not reduced.

[10] US Patent 3,693731 Sept 26 1972. Los Alamos National Laboratory. John S. Rowley.
See also http://www.osti.gov/scitech/biblio/7111588 and https://en.wikipedia.org/wiki/Subterrene (The notion of using the molten rock to form the tunnel lining is applicable, if at all, in low density soil or rock,-with sufficient void space to allow the molten rock to be accommodated by compaction of the rock i.e. without reducing the diameter of the tunnel. This is not practical in dense crystalline rock.)

[11] http://www.tunneltalk.com/AMR-India-Project1.php. See also https://www.youtube.com/watch?v=Qiqvtfmhj_4

[12] Wirth (now Aker-Wirth) has developed a later version of the CMM, the MTM (Mobile Tunnel Miner) http://www.youtube.com/watch?v=u6AJ-YvjbiA

[13] The depth of cut of the CMM discs in the Ruhr Sandstone trials was limited to avoid production of rock fragments too large to be accommodated on the CMM scraper and conveyor system transporting them away from the tunnel face.

[14] http://www.miningandconstruction.sandvik.com/__c125715e002ebf7d.nsf/Alldocs/s*5CContinuous*2Dmining*and*tunneling*machines*5CContinuous*reef*miners*2AMN220/$FILE/Brochure_MN220.pdf

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[16] Carly Leonida (2016) Making hard rock history. Mining Magazine http://www.miningmagazine.com/future-of-mining/future-of-mining-innovation/making-hard-rock-history/?adfesuccess=0

[17] CRCMining (2015) Oscillating disk cutter. http://www.crcmining.com.au/breakthrough-solutions/oscillating-disk-cutter/

[18] Handout notes. Gordon Revey, TAC –Rock Tunnelling Workshop, Sheraton Wall Centre Hotel, Vancouver BC November 16, 2013

[19] The applied fluid pressure in the hole will need to overcome in situ (compressive) tangential stress concentrations at the wall of will (assumed to be ~ 30MPa in this illustration) before the net tangential stress becomes tensile and the tensile (fracture) strength of the rock (ca 20MPa) is reached.

[20] The stress in the rock parallel to the long axis of the hole is also compressive as the detonation front progresses rapidly along the explosive column

[21] **Figure 9a** is from Persson et al; (1970).

[22] Fairhurst C (ca 1975) *Hard Rock Blasting Developments and Possibilities* Bench Drilling Days, Atlas Copco, Stockholm, Sweden. **Figure 4**.

[23] A rock blast dynamic simulation code developed by Itasca Consulting Group, Inc.

Biographies

Dr Sevda Dehkhoda is a Research Scientist at CSIRO Australia, working on mechanized rock excavation and breakage technologies for hard rock mining. Her core expertise is in rock mechanics and rock fracture mechanics. Sevda's research focuses on cutter/rock interaction and mechanics of rock failure in dynamically activated cutting tools specifically Actuated Undercutting Discs technology as well as alternative breakage methods such as high-pressure (pulsed) water jets. She aims to reduce the energy required for breaking hard rocks, improving the efficiency and efficacy of the excavation machines.

Charles Fairhurst, Professor Emeritus, University of Minnesota, Minneapolis, USA; Senior Consultant, Itasca International Inc. Minneapolis, obtained his Ph.D. in Mining Engineering from the University of Sheffield, UK in 1955. He joined the University of Minnesota faculty, School of Mines and Metallurgy in 1956, serving as Head for several years to 1970, when the Mining program was joined with Civil Engineering to form the Department of Civil and Mineral Engineering. He served as Head of the joint Department from 1973-87, and retired in 1997.

He has consulted on rock stability problems for tunnels, dams, mines and excavations throughout the world. He remains active in consulting, with a current emphasis on the mechanics of fracture propagation in naturally fractured rock and the effective stimulation of geothermal reservoirs. He served as President of the International Society for Rock Mechanics from 1991-1995, and has been elected to the U.S. National Academy of Engineering (1991) and the Royal Swedish Academy of Engineering Sciences (1979). He is a Fellow of the American Rock Mechanics Association.

Dr. Fairhurst holds honorary doctorate degrees from the University of Nancy, France; St. Petersburg Mining Academy, Russia; University of Sheffield, England; and University of Minnesota, USA; and is Advisory Professor to Tongji University, Shanghai, China.

In December, 2013, he was inducted as Officier, Légion d'Honneur, France.