PRIMARY ROCK EXCAVATION

Using the

IKC Impact Kerfing Cutter

(Excavation: rates, forces, and working life)

FINAL REPORT TO THE US DOE

January 2003

By Howard J. Handewith

**Report on the development of the RAMEX
Impact Mining Boom and Hard-rock Excavation System**

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 Principal Investigator

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BACKGROUND

The RAMEX excavation concept was developed and patented by Bill Coski. The patented concept consisted of a prototype Impact Kerfing Cutter (IKC) mounted on a high-energy Ram Mining Boom (RMB). The ram was boom-mounted on a crawler tractor and capable of excavating any shape of opening in rock or concrete.

The mining ram contained a massive diesel driven free-floating piston. When fired the piston would travel forward and transfer energy, through an air cushion, to the more

massive mining ram. The ram then created a high-energy impact at a comparatively low impact velocity. Rebound of the floating piston lifted the mining ram and cutting tool free of the rock after each blow. This action allowed free movement and repositioning of the mining boom without side loads or scraping of the cutting tool point(s) over the rock.

In contrast with the mining boom, a hydraulic breaker having the same impact energy and blow rate will require:

- Several tons of force to continuously hold the cutting tool to the rock. No lift-off between blows.
- The smaller piston to have over two times the impact velocity.
- The piston to impact the cutting tool with an uncushioned metal-to-metal contact.
- The breaker to impart severe side loads and high wear on the cutting tool.

A general objective of this research was to determine if the IKC could extend the useful function of the long proven hydraulic breaker by tunneling and trenching in rock. And to determine what effect, if any, the above items would have on the performance and operating costs of the IKC cutter.

The RAMEX RMB was conceived and patented to fill an increasing market driven need for a mechanical excavation tool to replace the explosives now in use for drift, or vein mining, and for the excavation of irregular shaped openings in rock or concrete. This need is especially critical for excavations in developed urban locations.

The original IKC was developed as a prototype-cutting tool for use on the RMB. Early field-test results of the IKC operating in a 40,000 psi (UCS) basaltic-andesite were encouraging. The initial production rates indicated that further development of the IKC cutter could be justified. However more work was needed to determine the cutter's economic operating life and evaluate its suitability for use on a hydraulic breaker.

The untimely death of the inventor, Mr. Bill Coski P.E., in 1990 required a re-evaluation of the Ramex program. Development of the IKC rock cutter continued but progress on the mining boom was postponed until the talents and resources necessary to replace Mr. Coski were acquired. Dr. G.T. Lineberry made an extensive evaluation of the potential of the Ramex mining system for NIST. As a result of his 1994 study Ramex was awarded an NIST/ERIP grant for assistance in developing the IKC rock-cutter. The specific objectives of this development effort were to:

- determine the suitability of the IKC for use on a conventional hydraulic breaker,
- optimize the IKC rock cutting efficiency,
- reduce the IKC manufacturing cost, to determine the operating costs and profitability, and to
- introduce the IKC for use to the mining and construction industries.

THE IKC ROCK CUTTER

A strong demand exists to eliminate the use of explosives in both the mining and construction industries. Underground the use of explosives causes lost time for; smoke and ventilation, lost time for the placing of additional ground support, and lost time for the additional handling of overbreak. In addition, the perceived hazard of handling and using explosives underground or in a developed urban area has caused much concern regarding the future use of explosives

At this writing most primary rock breaking in the industry is conducted with explosives. The exception, of course, is the tunnel-boring machine and its use is limited to long tunnels with circular cross-sections. As noted in the Appendix, (Handewith, November 2000) during the last twenty years many novel methods have been tested for primary rock breaking. Most of these developments have been found lacking in economic effectiveness or in practical application.

To break rock successfully, one must not only have an understanding of the rock fracture process but also the inherent operating economics. Hydraulic breakers have been used for secondary breaking operations for nearly thirty years yet they have little effect in primary rock breaking. (Handewith, 1980)

Rock Breaking Modes (Handewith, May 1990)

1. *Primary breaking* - initiating an opening or a slot in a rock surface (i.e., tunneling, trenching, drilling, boring, reaming, etc)
2. *Secondary breaking* - reducing the size of a free standing unit of rock or concrete (i.e., slab, boulder, erratic, etc.)

The mounted hydraulic breaker is a long proven and reliable secondary rock and concrete breaking machine. Chisel,moil, and blunt pointed tools mounted to the breaker are economical and effective for nearly all secondary breaking operations. Working in soft sedimentary formations, hydraulic breakers have had limited success with tunneling, the removal of tights, and for the excavation of special shaped openings. However, breaker technology has as yet to allow the primary breaking of rock. Existing tools are incapable of primary breaking for tunneling or trenching operations.

The patented IKC cutter is a logical development employing the kerf cutting concepts long used in the design of the raise boring machine (RBM) rolling button cutters. The rolling button cutter is widely used on RBMs for reaming both construction and mine shafts throughout the world. The RBM cutter consists of a rolling cone about 15-1/2" in diameter. It mounts either three or five disc rows, each of which is studded with tungsten carbide buttons. Rock chips break out between the button-studded rows in an action known as "kerf cutting." Kerf-cut rock chips are large compared with cuttings from any drill hole. Rock drill cuttings must be small in order to be flushed from the bottom of the

hole with air, water, or mud. Raise boring cuttings fall from the shaft with gravity and require little or no flushing action. Impact kerf cutting action with the IKC cutter is closely comparable to the rolling cutting action RBM cutter. The difference between the IKC and the RBM cutter is that the RBM cutter applies a rolling thrust load into rock, similar in action to that of a gear tooth. Whereas the IKC applies repeated impact force loads and it is moved over the rock surface by a boom mounted on a hydraulic excavator.

General Principals of Rock Breaking

Primary Breaking

- *A surface of a rock must be stressed for it to fail. That stress is a function of the tool geometry and applied force. Generally this stress is in the range of 100,000 to 150,000-psi. The contact stress is highest when failure is initiated.*
- *A single point impact will produce a conical indentation in a rock surface. Future single point impacts will tend to enlarge that cone.*
- *Large rock cuttings require less energy to create. It follows that tools producing larger cuttings will be more efficient, experience less wear, and have a longer life. It follows that they should also be more economical.*
- *Cuttings must be flushed from under a cutter to prevent excessive, wear, and energy loss caused by regrinding.*

Secondary Breaking

1. *Secondary breaking is caused when a crack is initiated from one free surface and grows to second free surface. The two surfaces must intersect at an angle greater than 90°*

Though both the RBM and IKC cutters cut 'kerfs,' the IKC has an additional feature in that it can also cut slots. It can leave large cores of rock between the kerf cut slots and these cores can be broken out using secondary breaking methods. Impact Kerf Cutting comes into its own, however, when one looks into the energy required to break out the rock cores left standing free between the kerf cut slots. The IKC can cut slots up to 18" deep. Large rock cores can be left between these slots where the machine operator can determine the core width by experimentation. These cores can be broken using either a conventional moil point tool or the IKC cutter. This "core breaking" operation produces very large rock cuttings, often in the range of minus ten inches.

A single point impact will produce a conical shaped crater in rock (Dollinger et. Al., 1998). Repeated impacts using a single point tool will simply enlarge the cone shape to

the point where the tool is simply deflected from the inside surface of the cone. The single point tool is incapable of excavating a slot. A unique and patented feature of the IKC cutter is its ability to kerf-cut slots as shown below.

IKC Cutter for 9" Wide Slots

Section Through an IKC Cut
Trench or Tunnel Face
(Three slots and two cores wide)

Tungsten Carbide Buttons • a Key Element of the IKC Cutter

The mining and construction industries have employed the use of tungsten carbide buttons for impact excavation over the last forty years (Handewith July 2000). All buttons used in this development were made up of mining grade tungsten carbide as currently supplied by Kennametal. They have a hemispherical top and are 3/4" in diameter. They extend 3/8" above the cutter surface.

A Tungsten Carbide Button

To compare the laboratory drop tests with the field testing it was elected to use the 'blow-force-per-button' as a common denominator. The objective of the laboratory drop tests at the US Bureau of Mines Mining Research Center was to evaluate any influence higher button cutter loads would have on IKC production performance. (Haselton, October 1995)

HISTORIC LOADING OF TUNGSTEN CARBIDE BUTTONS

	Use of Tungsten Carbide Button	Average Force/Button	Remarks
1	IKC Prototype Ct'r	105 ft-lb	5,500 ft-lb/blow / 52 buttons
2	BOM Drop tests	250 ft-lb	Single button drop test
3	IKC on Hyd. Braker	111 ft-lb	3,000 ft-lb / 27 buttons
4	RBM Rolling Cutter	800 - 1,500 lb	Rolling Force Load
5	In-The-Hole Hammer	600 to 800 lb	Up to 36" Diameter Impact Cutters

The IKC cutters used at the Cadman Quarry had an average loading of about 110-lb per button. The low tungsten carbide button load was used on the prototype cutter to provide a maximum tool life and minimize any cutter-caused down time on the development operation. The Bureau of Mines Drop Testing was designed for cutter force loads of 250 ft-lb/button to investigate the penetration of a tungsten button with a less conservative force per blow.

When mounted on the impact-mining boom the IKC cutter lifted off the ground, up to 1-1/2 inches, following each blow cycle. When mounted on the hydraulic breaker the cutter

remained in constant contact with the rock throughout the entire blow cycle. For this reason it was expected that the IKC might show more wear when mounted on a hydraulic breaker. Another objective of this development was to investigate what effect not lifting-off-of-the-rock after each blow would have on the life and the production or wear rate of the IKC.

TEST PROGRAM

PHASE I Cadman High Rock Quarry Testing (1988-1990)

Coski Enterprises Ltd., designed and built both a diesel driven Ram Mining Boom (RMB), and two IKC rock cutters. Ramex Systems (Canada) later joined Coski Enterprises in this effort as a full partner. (Handewith, June 1989)

Initial testing of the IKC cutter was conducted on the full-scale prototype diesel driven IMM. Testing was conducted at the Cadman quarry situated near the City of Monroe Washington. This quarry is noted for producing durable riprap and landscaping stone. Quarry rock is a hard and abrasive basaltic-andesite with an unconfined compressive strength in the range of 25,000 to 40,000-psi. In some areas the rock was also fractured due to prior blasting. In general the rock had a close set moderately tight joint system spaced at 4 to 8 inches.

To insure that cutter failures did not plague field-testing, the prototype IKC cutter was conservatively designed with a low force load (125 ft-lb) on the buttons. It was to last throughout the development operation. Maximum working life was the primary consideration with rock cutting efficiency as a secondary consideration. The IMM had a total of 100 operating hours when operations were shut down in early 1991. About 50% of the operating time was classed as 'full-load testing.' Two cutters were used over the prototype test period. The first failed when one of its 52 tungsten carbide buttons sheared off due to improper handling. This occurred after about 30 hours of operation. Though the cutter was usable, it was thought that the missing button could influence the production rates so it was removed and used as a spare. The second cutter completed the additional 65 hours of testing.

Once the load and blow rate capabilities of the RMB was verified, an air/water-mist sub-system was tested to flush cuttings from under the cutter. This was done to minimize the inefficient regrinding of the rock cuttings. About two gallons a minute of water was injected into the 120-psi (at 250 scfm) air system. This proved effective at removing rock cuttings from under the cutter. It was however, limited in its ability to remove airborne respirable dust. As reported by Kovscek, et. al., in 1991 (see Appendix) the mitigation of airborne dust needs more work. It should be noted that this is a problem with all hydraulic breaker operations and is not specific to the IKC cutter.

PHASE II Laboratory Drop Testing

The Phase II Laboratory Drop tests were conducted in 1994 and 1995 under the auspices of the NIST/ERIP grant in conjunction with a cooperative effort from the University of Washington and the US Bureau of Mines. Mr. H. Haselton (1995) reported these tests as part of his studies for a Masters of Science in Civil Engineering at the University of Washington. (See Appendix)

The purpose of this testing was to determine any performance difference between the high velocity-low mass impact of a hydraulic breaker and that of the IMM with a low velocity-high mass. The difference being that a hydraulic breaker has an impact velocity of about 20 ft/s while the IMM's was only 10 ft/s. A second objective was to determine the effect of using a blow force of 250 ft-lb per button in place of the conservative design of 125 ft-lb used on the IMM at the Cadman quarry. A final objective of this effort was to develop performance-estimating criteria for the IKC in several types of rock and concrete. This was to include investigating any influence that the energy force-rate delivery ability of the several different size hydraulic breakers would have on operating performance. (Handewith, 1968)

To accomplish these objectives without incurring the expense and uncertainty of full-scale field testing, a series of laboratory drop tests was scheduled to take place at the Mining Equipment Test Facility of the US Bureau of Mines in Pittsburgh. Drop tests were conducted on large samples of:

1. standard strength concrete with a UCS of 3,600 psi
2. high strength (9,900-psi) concrete with a UCS of 9,900 psi
3. Berea sandstone with a UCS of 6,700-psi and,
4. Barre granite having a UCS of 28,600-psi

The general finding was that impact velocity had little detectable effect on rock penetration and that the high velocity hydraulic breaker could be expected to perform equally as well using the IKC as it did on the original low velocity prototype IMM.

PHASE IIIA - Testing of the re-designed IKC on a 3,000 ft-lb Hydraulic Breaker

Full-scale testing was conducted at the Abbotsford quarry on February 26th and 27th 1997. The quarry is situated on a massive greenstone outcrop just outside the city of Abbotsford BC in Canada. This testing was conducted to demonstrate operation of the newly re-designed IKC cutter. This new cutter was 9" wide by 16.5" long and supported 27 face and 4 gage buttons. (The original cutter was 16 inches square and mounted 52 carbide buttons.) The hydraulic breaker was an NKP H-10XB with blow rating of 3,000 ft-lb. The cutter button force-load was calculated to be the same as observed at Cadman in the range of 110 to 125 ft-lb per button. The breaker was mounted on a Komatsu PC200-SL 45,000-pound hydraulic excavator. Testing was conducted on an undisturbed outcrop of greenstone with an exposed surface angled about 30° above vertical. The angle

of the slot (above the angle of repose) allowed gravity to clean cuttings from under the cutter and flow freely from the slot. A series of 9-inch deep sump tests were conducted for repeatability. It was found that the breaker was operating at a rate of 464 blows per minute and mining rock at an average rate of 2.4 cubic feet per minute. This calculated to an average IKC sump rate of 0.06 inches per blow and a production rate of 5.3 cubic yards (13.3 tons) per hour.

The IKC used consisted of two parts; a flat plate cutter mounting 27 buttons was bolted to an upper shaft. Periodic loosening and breaking of the attachment bolts was a problem and it was elected to use a single unit design for future testing. Other than the bolts, the cutter showed no wear after eight hours of operation over the two-day period. Test results were good enough to encourage further testing.

PHASE III B Production Testing with a 3,000 ft-lb Hydraulic Breaker at Atkinson Property, West Vancouver, BC Canada 1998

ERIP program funding was nearly exhausted at this stage of development and Ramex elected to continue the program by undertaking a project that could offer some economic return. The Atkinson family, residing in a wealthy suburban area of West Vancouver BC Canada was building a new home that required the excavation of some 700 cubic meters of granite. The city council prohibited the use of explosives and the only alternative available to the family was some form of mechanical excavation. A contract was signed with the Atkinson family to excavate some 700 cubic meters of granite.

The property is situated in coarse grain granite as represented by the West Vancouver pluton in BC Canada. As planned a 6,000 ft-lb hydraulic breaker mounting the newly built single unit IKC cutter would be used. This breaker could deliver an average blow force of 250 ft-lb per button allowing a direct comparison to the laboratory drop test data reported by Haselton 1995. However, financial considerations limited the project to use of the same 3,000 ft-lb hydraulic breaker used at the Abbotsford quarry. The smaller breaker provided an average cutter button force load of about 125 ft-lb providing a comparison with the previous field tests at Cadman and Abbotsford.

For the same economic reasons stated above only one IKC cutter was available for this project. After eight hours of operation the upper end of the cutter shaft failed rendering the cutter useless for further testing. Such failures were common in early breaker tools but rarely occur today. Work was continued using a conventional moil point tool. This operation reaffirmed that it is not practical or economical to use a relatively small 3,000 ft-lb hydraulic breaker and the inefficient moil point tool for primary excavation of granite.

It was estimated that the IKC mining rate, prior to the failure, was in the order of 3.6 cubic feet per minute (calculating to a sump rate of 0.09 inches per blow). Working a full 60-minute hour provides a production rate of 8.0 cubic yards per hour (17.5 tons/hour). Based on the laboratory drop tests it is expected that these production rates would have doubled to 16 cubic yards per hour using a 6,000 ft-lb hydraulic breaker and a 27-button IKC cutter.

Moil point excavation did produce some additional useful information regarding muck removal, dust suppression, noise abatement, and ground vibrations as indicated in the attached Appendix written by Copal, 1998.

PRODUCTION RATE ESTIMATING

The laboratory drop tests were conducted using an average tungsten carbide button load force of 250 ft-lb. All three field tests, one with the 16 inch square IKC cutter mounting 52 buttons, and two using the 9 inch wide IKC cutter with 27 buttons had an average force per blow ranging from 110 ft-lb to 125 ft-lb. Specific energy and statistical studies indicated that it is reasonable to assume a linear relationship of force to penetration over the range of 125 to 250 ft-lb / blow / button. The penetration of a button loaded to 225 ft-lb should reasonably be twice that of a button loaded to 125 ft-lb.

The original 16-inch square IKC was designed by sacrificing cutting efficiency for operating life. Even so, the cutter performed well at the Cadman Quarry where it demonstrated production rates ranging from a low of 9.4 insitu cubic yards per hour (22.2 s Ton/hr) to a high of 18.5 insitu cubic yards per hour (44 s Ton/Hr). The higher excavation rates were achieved when the air/water mist system was working, preventing any production rate loss due to regrinding of the cuttings.

SPECIMEN	UCS (psi)	Tensile (psi)	Young's Modulus (psi)	Density (pcf)	Schmidt Rebound R Value	IKC Penetration (inches / blow) ⁽⁵⁾	
						250 ft-lb	125 ft-lb
Low Strength Concrete ⁽¹⁾	3,620	360	3,000,000	135	39	0.33	0.16
High Strength Concrete ⁽¹⁾	9,915	572	5,000,000	142	52	0.15	0.07
Berea Sandstone ⁽¹⁾	6,700	155	2,000,000	132	52	0.34	0.17
Barre Granite ⁽¹⁾	28,600	1,110	6,600,000	165	70	0.08	0.04
Basaltic Andesite ⁽²⁾	32,000	2,600	7,500,000	178	62	0.08	0.04
Greenstone ⁽³⁾	21,500	1,400	6,000,000	186	46	0.12	0.06
W. Vancouver Granite ⁽⁴⁾	11,300	790	5,800,000	174	37	0.18	0.09

from an estimate of the observed production rate.

⁽⁵⁾ All field test data was based on an average button load of 125 ft-lb. The 250 ft-lb penetration is presumed to be twice that of the 125 ft-lb penetration.

One objective of this investigation was to establish a production rate estimating procedure. A linear regression study was made of the physical properties of samples from the laboratory and field-testing. Test results are shown above and in the attached Appendix as "IKC_Predictor.Xls" UCS, tensile strength, Young's modulus, and specimen density values were paired with the IKC penetration at force load of 250 ft-lb per blow. Schmidt Hammer R value was then paired to both

STATISTICAL RANKING OF IKC TEST DATA R^2

4) (R x pcf) as an indicator of Unconfined Compressive Strength	0.92
3) Rebound Value as an indicator of Unconfined Compressive Strength	0.54
7) Young's Modulus as an indicator of IKC penetration	0.92
1) UCS as an indicator of IKC penetration	0.75
2) (UCS / pcf) as an indicator of IKC penetration	0.74
6) (Rebound x pcf) as an indicator of IKC penetration	0.66
8) Tensile Strength as an indicator of IKC penetration	0.58
5) Rebound value as an indicator of IKC penetration	0.31

This statistical ranking indicated that:

1. The Schmidt Concrete Test Hammer R value, when modified by the specimen density in pounds per cubic foot (pcf), is a reasonable indicator of rock uniaxial unconfined compressive strength (UCS). This conclusion is supported by previous work reported in the literature by Deere & Miller 1965, Aufmuth 1972 and Reichmuth 1963.
2. Young's modulus is the best indicator of IKC penetration
3. That the Uniaxial Unconfined Compressive Strength is a reasonable indicator of IKC production rate.
4. IKC penetration per blow can be calculated within reasonable limits using the pcf modified Schmidt Hammer test data to calculate UCS. Then UCS can be used to estimate IKC penetration.

Estimating IKC performance is predicated upon knowing the operating characteristics of the hydraulic breaker (such as force/blow and blow rate) along with an understanding of the physical properties of the rock to be excavated. The most critical rock property is its rate of deformation under a known load. Three estimating methods were developed and they are listed below in order of descending reliability: (Handewith 1980)

1. Mobilizing a field test using a hydraulic breaker and undercarriage system working in the project rock to determine the actual IKC penetration, or deformation rate.
2. A more cost-effective method of estimating excavation rate is to perform a laboratory test for both UCS and Young's Modulus and calculate the rock deformation based on the above table "Summary of field and laboratory tests."
3. A third alternative for estimating performance uses the Schmidt 'L' Test Hammer procedure outlined by Deere and Miller (1956) value times the specimen density in pcf to determine the UCS value. The UCS can then be used to estimate IKC penetration as outlined in procedure 2 above.

It should be noted that all IKC rock penetration data is for excavating kerf-cut slots and that the actual production will be 25% to 50% faster when the core-breaking operation is accounted for.

ESTIMATING KERF-CUT SLOT PRODUCTION RATES* USING THE IKC CUTTER

SPECIMEN	Cubic Yards per Hour (in situ)	
	Hydraulic Breaker	
	6,000 ft-lb at 400 bpm	3,000 ft-lb at 460 bpm
Low Strength Concrete	25.80	14.39
High Strength Concrete	11.73	6.29
Berea Sandstone	26.58	15.28
Barre Granite	6.26	3.60
Basaltic Andesite	6.26	3.60
Greenstone	9.38	5.39
W. Vancouver Granite	14.07	8.09

- These rates should be 25% to 50% faster when breaking rock cores.

SUMMARY and CONCLUSIONS

- The first program objective was to determine the suitability of the IKC cutter for future use on a hydraulic breaker. The concept was to extend the proven capabilities of a hydraulic breaker into the realm of rock excavation and trenching. (This is now exclusively the realm of explosives.) Production rates achieved while excavating rock proved to be competitive with those of explosive excavation. However the IKC tool life was too short and the resulting cutting tool costs were too high to justify further development of the IKC as a tool for use on a hydraulic breaker. The reason was that the hydraulic breaker is a delivery system requiring its cutting tool be crowded into rock contact at all times. That crowding force has to be sustained even while the boom is being repositioned. And repositioning the boom while the cutting tool is loaded causes the buttons to be scraped over the rock resulting in high side loads and premature button failure.
On a more positive side, it was reaffirmed that the reduced cost IKC cutter was economical for operation on the Ramex Ram Mining Boom (RMB). The RMB action lifts the cutting tool from the rock following each impact cycle. This action allows the RMB to be repositioned after each impact without penalty on IKC operating life.
- The 27 button IKC field-tested in granite performed as well as the 52 button IKC used at the Cadman Quarry when it had the same average impact force per button. This indicated that the 27 button IKC was a much more efficient rock-cutting tool.
- The manufacturing costs of the IKC cutter were reduced by nearly 50%.
- Test data clearly indicated that the linear-elastic material properties of rock normally used for design of underground structures are insufficient to predict how and at what rate rock will fail when it occurs in a thick and unconfined plate such as in tunneling or trenching. However there is a strong indication that the coefficient of restitution may be the single rock and rock-mass property that could provide accurate failure rate predictions. It is recommended that a new program be undertaken to assess, both in the laboratory and in the field, the rock and rock mass coefficient of restitution and to establish what, if any, relationship that property has to rock failure (fracture) rate prediction.
- The Schmidt Concrete Test Hammer is a reasonable predictor of uniaxial unconfined compressive strength of rock. It is a much better predictor when compared to the product of the Schmidt R-Value times the density of rock in pounds per cubic foot. (Deere and Millie 1956 and R. Aufmuth, 1972)

- It was established that the projected surface area of a cutter is proportional to the cutters production at the same blow force level (Cadman Rock IKC and Abbottsford IKC)
- The fact that a hydraulic breaker did not lift the IKC cutter off of the ground after each blow cycle appeared to have a detrimental effect on the IKC tool life.
- Although the IKC operating life was not firmly established it was determined to be much in excess of 20 operating hours.
- The pressurized air/water system was effective at cleaning under the cutter preventing re-grinding of the rock chips.

The following items are common to those found on operating hydraulic breakers today. It is incumbent upon the breaker manufacturers to address these areas of concern.

Muck pick up and removal is a problem in any trenching operation. This is an area requiring more work by Ramex for ongoing development of the IKC cutting tool.

Dust Suppression

The Ramex pressurized air/water mist proved effective at cleaning from under the IKC cutter, but it had little effect on the generation of airborne dust. An effective spray system needs to be developed.

Noise Abatement

Noise of the IKC operating on a hydraulic breaker is the same as the noise of a breaker operating with any standard tool.

Ground Vibration

Limited ground vibration data was taken from the IMM operation at Cadman and from the 3,000 ft-lb breaker working in West Vancouver. Again these vibrations appear to be the same as those of a standard moil point tool working in the same rock.

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**Research on the ongoing development of the Ramex impact cutter for
rock excavation**

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I wish to dedicate this paper to my dog Spunky, my constant companion during much of the writing of this document, who constantly reminds me to live for the day, sleep in the sun when possible, and never to take life too seriously.

1 Introduction

1.1 Introduction

This report documents part of the continuing research, development, production, testing and application of the patented Ramex rock excavation system.

The Ramex system is a method of excavating medium to very hard confined rock using a patented Impact Kerf Cutter (IKC). The impact energy is provided to the cutter by a Ramex patented diesel ram or a conventional hydraulic hammer (see Figure 1). The cutter and hammer are mounted on a suitable undercarriage capable of positioning the cutter against the in-place rock. The cutter simultaneously impacts the rock in more than one location through kerf cutting rows of hemispherical tungsten carbide buttons (see Figure 2). This impact generates fractures in the rock beneath the buttons. The coalescence of these fractures causes the rock to fragment into chips that can then be removed. By sweeping the cutter across the face, essentially flat sided slots can be excavated in rock. Figure 3 shows a rock face after the cutter has been swept down a rock face multiple times. Excavation of a near flat sided slot is difficult if not impossible for most methods of rock excavation.

Figure 1 Ramex system with hydraulic hammer.

Figure 2 IKC showing tungsten carbide buttons.

Figure 3 Slots in rock excavated by Ramex cutter.

The most efficient method of operation is to excavate slots, in the solid rock (see Figure 4). If the slots are properly spaced, the remaining unconfined sections between adjacent slots quickly break into much larger fragments using much less energy. The system is unique in allowing the operator to exploit zones of weakness in the rock mass and take advantage of nonhomogeneous rock.

Figure 4 Ramex method of excavating on a face showing development of confined slots and unconfined core sections.

Applications of the Ramex system include most any type of excavation in medium to very hard rock, such as the following: trenching, general excavation, tunneling of any shape, portal development prior to use of a Tunnel Boring Machine (TBM), cross shafts and utility openings off of TBM excavations, mining of all types of openings, and civil demolition work. One of the biggest potential uses for the system is in areas where the use of explosives is either unlawful or impractical, such as excavations near an existing structure. The Ramex system is one of the few alternatives available to excavate medium to very hard rock without the use of explosives.

1.2 History

The Ramex rock excavation system was conceived and developed in the 1980's by the late William Coski, PE. The concept was developed and patented with a million dollars in private funding. After initial testing in 1989, Ramex received a grant from the Department of Energy to develop the IKC, and has since been working with the University of Washington to continue research and development.

The initial prototype system was tested in 1989 and 1990 at the Cadman quarry near Monroe, Washington. At the Cadman site, the IKC was connected to a Ramex-designed prototype low velocity/high mass diesel-powered ram. The focus of the research over the last several years has shifted away from the diesel ram to the adaptation and development of the concept using conventional, high velocity/low mass hydraulic hammers.

This paper focuses on the period of research that includes the initial trial tests of the cutter connected with a conventional hydraulic hammer, through the first successful commercial application.

2 Literature Review/Background

2.1 Rock Breakage Using Impact

Previous Ramex-sponsored research includes an excellent literature review concerning both static and dynamic confined rock fracture [Haselton 1995]. Research by other investigators on rock breaking by impact includes a study on fragmentation by indentors [Tan, Kou, and Lindqvist 1996]. In this study, a simulation model was presented that refined the theory of crack propagation around an indenter. This research also discussed crack coalescence between adjacent indentors (see Figure 5). The study also suggested that hemispherical shapes, as used in the Ramex IKC, are more efficient than truncated indentors (see Figure 6). Whittaker et al. (1992) also studied the coalescence of cracks between adjacent indentors. Detournay et al., (1995) studied the formation of a crushed zone and fractures that occurs beneath a wedge. The study of impact loading on semi-infinite brittle materials [Goldsmith, 1960; Fuchs, 1963; Liaw, 1983] found a threshold stress existed that must be exceeded to produce fractures.

Figure 5 Crack coalescence between adjacent indentors.

hemispherical

truncated

Figure 6 Indenter shapes.

2.2 Energy Considerations

For any method of rock excavation, energy is required to fragment the rock into material that can be easily removed. Applied energy that only compresses the rock and does not produce crushing or fractures is useless for the excavation process. However, any applied energy that crushes rock finer than necessary for removal is unneeded and wasted. The minimum specific energy for excavation requires breaking the rock into the largest possible pieces that can be easily removed from the intact rock mass, thereby minimizing energy used to fracture the rock [Bond 1961]. Specific energy is the energy consumed per unit volume of rock. The surface area of the broken rock is directly related to the area of the fracture surfaces, and thus to the energy required to produce these fractures. Existing natural fractures in the rock are fracture surfaces that do not need to be made by the cutter, therefore less energy input is required for a rock mass with more fractures. The specific energy decreases with increased input energy [Grantmyre and Hawkes 1975, Wayment and Grantmyre 1976]. The relationship is approximately as follows:

$$Esp = K/\sqrt{(Eb)}$$

Esp = Specific Energy

K = Constant

Eb = Energy per blow

This relationship shows that the largest practical impact will lead to greater efficiencies, producing larger rock fragments. These principles are essentially the same for crushing, blasting, drilling, and other mechanical methods of rock excavation, including TBM's.

2.3 Hammer And Tool considerations

Ramex developed and patented a diesel ram or hammer (see Figures 7 and 8) that works in combination with the separately patented IKC. The prototype diesel ram incorporates a 100 design hp free floating piston that operates in line with the IKC. The diesel ram was designed for 5500 ft-lbs/blow with 5 kerf cutting rows. The literature states that the ram is capable of 700 to 800 blows per minute. The diesel ram produces a low velocity (10 ft/sec), high mass impact with the rock, compared to the high velocity (20 ft/sec), low mass impact of a hydraulic hammer. Much of Ramex's initial research and development focused on the diesel ram. Handewith, Coski & Thimons (1989) discussed development and testing of the diesel ram. The diesel ram was not operated or tested in the current research, as this research focused on development of the IKC in combination with conventional hydraulic

hammers. The decision to investigate the Ramex method in combination with the hydraulic hammer was based on previous research by Ramex and the University of Washington [Haselton 1995]. The research indicated the IKC could be adapted for use with a hydraulic hammer. This conclusion was reached by laboratory testing of a simulated Ramex cutter in a simple drop test apparatus on rock and concrete specimens using a constant energy level and different combinations of mass and velocity.

Figure 7 Ramex patented diesel ram.

Figure 8 Ramex patented diesel ram schematic diagram.

Hydraulic hammers are high energy impactors that were developed over 30 years ago and have since undergone many refinements [Wayment and Grantmyre 1976]. Figure 9 shows a typical hydraulic hammer schematic diagram. They are currently produced by a number of manufacturers. Available hammer impact energies range from 500 to 20,000 ft.-lbs/blow. Typical blow rates are around 500 blows per minute for hammers in the 3000 ft.-lbs/blow range. Uses include secondary breaking, demolition, and mining. Hydraulic hammers are typically mounted on conventional hydraulic excavators or backhoes in place of the bucket, and are occasionally mounted over rock crushers or grizzlies. Power is supplied by the undercarriage's hydraulic system. The hammers are currently fairly reliable, commonly available, and easily connected to standard excavators or backhoes [Bauer 1994].

Figure 9 Typical hydraulic hammer schematic diagram.

The hydraulic hammer output energy can typically be varied by changing the internal charge pressure or selecting a different hammer. Since the charge pressure also varies due to heat, it must be checked after the hammer is warmed up. For all testing involved with this research, a Nippon Pneumatic Manufacturing Co. (NPK) hammer with a rating of 3000 ft.-lbs/blow was used with the charge pressure at the manufacturers maximum recommended pressure (see Appendix A for hammer specifications). NPK is a Japanese company that manufactures hydraulic hammers that are commonly available in the U.S. and Canada.

The typical tools used on hydraulic hammers are cylindrical with a moil, blunt, or chisel shaped ends (see Figure 10). However, after some use, the points of the tools tend to wear to approximately the same blunt shape, regardless of the initial machined shapes. Excavation in hard rock that is confined, such as tunneling or trenching, with any of the standard hydraulic hammer tools is difficult if not impossible. These tools tend to shape the working face into a cone where no free surface is available for the rock to break to [Handewith, Coski, and Thimons, 1989]. Excavation in hard rock is possible with the standard tools as long as a free surface is available for the rock to break to.

Figure 10 Typical hydraulic hammer tools.

The tools connect to the hammer via a cylindrical shaft, with diameters typically ranging between 2 and 12 inches. The upper end of the round shaft is machined to key into the hammer to allow for changing or replacing tools. Changing tools is typically a relatively quick process. Unfortunately, tool compatibility is not standard among hydraulic hammer manufacturers, and in most cases, tools are not even interchangeable between hammers of different rated energies produced by the same manufacturer. The IKC can be machined to connect to any manufacturer's hammer; however, it must be machined to fit only one specific hammer. This makes testing the IKC using hammers with different energies or hammers of different brands difficult and expensive.

The IKC used for this research (see Figure 2 and 11) had a shaft diameter of 5 inches with a total of 32 tungsten carbide buttons. Twenty two buttons are aligned in three main kerf cutting rows spaced 3 inches apart, and ten buttons are around the perimeter to cut the gage or sides of the slot. The cutter was

designed and machined to connect to a 3000 ft.-lbs/blow NPK hydraulic hammer.

Figure 11 IKC mounted on a NPK-10XB hydraulic hammer

2.4 Vibration considerations

One of the advantages of the Ramex method is the minimal disruption to nearby structures, compared to the disruption associated with blasting. From tunneling experience in Italy, Fornaro et al. (1993) conclude that peak vibration due to excavation using hydraulic hammers with a standardmoil point is about an order of magnitude less than peak vibration from blasting, and further, a TBM produces about an order of magnitude less vibration than a hydraulic hammer. Vibration from the IKC is likely the same as produced by amoil point, assuming input energy is the same.

2.5 Rock mechanics considerations

The excavation rate using the Ramex method is very much dependent upon the characteristics of the rock. Rock can be tested and characterized in many different ways. Some classifications and tests may be relevant as a predictor of excavation rates, while some may not. The characteristics of rock that were considered to most strongly affect the IKC excavation rate were the average strength, and the orientation and spacing of existing discontinuities.

Armando et al. [1994], suggested that field seismic velocity measurements are the most practical way of obtaining data on average rock conditions as a predictor of tunneling rates. Their conclusions were based on tunneling experience in Italy using hydraulic hammers. Field seismic velocity testing takes into account in situ fracturing, as well as the average hardness of the rock. The velocities of seismic waves travel faster in stiffer materials and slower in softer materials or discontinuities. Field seismic velocity measurements have also been used successfully and universally for prediction of rippability of in-place rock. Seismic velocities determined from lab samples are significantly different from field seismic velocities, as lab testing of seismic velocities does not take into account the in situ discontinuities in the rock mass. Due to budget constraints, no seismic velocity testing was performed at the Abbotsford or the Atkinson sites discussed in this paper.

2.6 Cuttings removal considerations

A difficult part of any type of mechanical rock excavation method is efficiently removing the rock fragments produced. This is true for the Ramex method as well. Maximizing the efficiency of the IKC requires removing loose rock fragments from beneath the cutter, so energy from the IKC is transmitted directly to the unbroken rock. Any material trapped beneath the cutter is further crushed or broken by successive impacts of the cutter, wasting time and energy. Clearing the cutter maximizes excavation rates, decreases energy use, and minimizes cutter wear. Methods of clearing the cutter include pressurized air or water, mechanical, vacuum, or gravity.

The patented diesel ram and cutter prototype shown in Figure 7 had a cuttings flushing system consisting of mixed pressurized air and water. The air and water mix was forced out of the face of the upper part of the cutter in a downward direction. The system was designed not only to clear broken rock, but also to minimize dust. During the limited testing performed with this prototype, the flushing system was successful in accomplishing both

objectives. An air/water flushing system was also incorporated into the first IKC mounted on a hydraulic hammer. This flushing system failed after a short period of usage, due to breakage of a fitting on the cutter head, where the flexible supply hose connected to the IKC. This breakdown occurred prior to the current research.

3 Abbotsford Testing

3.0 Introduction

The Abbotsford quarry site shown in Figure 12 was used on February 26th and 27th, 1997, to test the first adaptation of the IKC to a conventional hydraulic hammer. The cutter had previously been used only with the Ramex patented diesel hammer. A cutter was designed and manufactured specifically to connect to a 3000 ft.-lbs/blow NPK hydraulic hammer. The hammer was mounted on and powered by a standard Komatsu excavator platform. Appendix B provides specific details of the equipment used. If the adaptation was successful, a comprehensive test program would be developed.

Only limited testing was done at this site due to budget restrictions and safety and space constraints of working in an active quarry.

Figure 12 Testing at the Abbotsford quarry site.

3.1 Geology / Site Conditions

The Abbotsford quarry rock is predominantly greenstone. The quarry is located just north of the town of Abbotsford, British Columbia. The strength of the rock varies from 18,780 to 53,800 psi, based on Schmidt hammer testing (as described by Miller [1965]). The density of the rock was measured at 185 to 187 pcf. The Schmidt hammer measurements and the density tests were done on representative samples and locations in the immediate vicinity of where the IKC was tested.

The quarry was in active production at the time of testing. The IKC was used primarily on a near vertical quarry face where previous blasting likely fractured the rock considerably. The IKC was also used on a 10 foot diameter boulder with no visible fractures, and on a protruding outcrop to simulate trenching excavation.

3.2 Testing

The equipment was operated for about 8 hours total. The IKC operated reliably during the testing. The operator noted an increase in production as he gained operating experience.

Excavation performance testing was done with the equipment operating for 40 minutes on a near vertical face. As previously mentioned, this site had been blasted, and some of the rock was fractured and broke apart fairly easily. The excavation rate was visually estimated at approximately 12 in situ cu-yd/hr. Adequate equipment was not available to weigh or measure the amount of excavated rock. Testing the equipment at this location likely resulted in an excavation rate higher than if the excavation had been done in an area that was not fractured by production blasting. Excavation rates were not estimated on the boulder or on the near horizontal outcrop due to time and access limitations.

The cutter was inspected periodically to determine cutter wear or damage. After 8 hours of use, the tungsten carbide buttons had no visible wear, while the steel in the cutter head exhibited slight wear between the buttons.

3.3 Conclusions

This short preliminary test indicated that the adaptation of the IKC to a hydraulic hammer was successful; therefore, further development and planning for a more comprehensive test program could be justified. The cutter

operated reliably with minimal wear after about 8 hours of total use.
Experience gained at this site was useful in setting up a more comprehensive quantitative test program.

4 Testing program

4.0 Introduction

Considerable effort and time went into developing a test matrix designed to isolate several important variables related to the production rate of the equipment, based on experience gained through initial testing at the Abbotsford quarry site. A testing program was developed but was not used due to limitations of budget, equipment, and availability of a suitable test site. Subsequently, a limited, non-ideal test program was fit in around a production schedule at a second site near Vancouver, British Columbia. This was a residential site belonging to the Atkinson family.

4.1 Planned testing

A test matrix was set up to isolate several variables independently (see Table 1). The purpose of the planned testing was to compare the IKC to the standard moil point cutter in a variety of conditions. After completion of the planned test program, Ramex would be better able to determine if any advantage existed over the standard moil point, and be better able to predict the performance of the Ramex method. Also, an assessment could be made of changes that could increase productivity of the Ramex method in the future.

		Vertical Face	
		Rock A	Rock B
Hammer 1	IKC	test 1	test 5
	Moil point	test 2	test 6
Hammer 2	IKC	test 3	test 7
	Moil point	test 4	test 8

		Horizontal Trench	
		Rock A	Rock B
Hammer 1	IKC	test 9	test 13
	Moil point	test 10	test 14
Hammer 2	IKC	test 11	test 15
	Moil point	test 12	test 16

Table 1 Planned Test Program

The planned test program consisted of 16 basic tests. The tests were to be done with two cutters-the IKC and the standard moil point. Two rock faces, horizontal and vertical, would be tested, using hammers of two energies. The tests would be performed on two different rock types. From this research, the advantages and disadvantages of either cutter could be determined. The research was also designed to produce several points on a graph that had three variables, input energy, seismic velocity, and production rate. The data from this research as well as data from further testing could be used to help predict production rate. Other variables not included in the test matrix would also be investigated including a cutter flushing system and a cuttings removal system.

4.2 Actual testing

The actual testing program differed significantly from the planned program described above. The Atkinson site was not ideal, primarily due to budget and time constraints associated with fitting the tests in to a production operation. Only one hammer was used throughout the testing due to the costs involved with renting a second hammer and machining a new cutter to fit (recall the cutters are not interchangeable). Further, the site contained only one rock type.

The actual testing program did produce useful results as discussed in the following sections. The test program was used to determine the production rate of the Ramex method on a vertical face, hammer blow rate, and cuttings gradation. Two types of cuttings flushing systems were also experimented with. The final testing program that evolved is described in detail in the following sections.

5 Atkinson Site

5.0 Introduction

The Atkinson project was the first commercial application of the Ramex IKC system. Ramex Corporation was selected as the excavation subcontractor in the spring of 1997 for construction of a private residence located in West Vancouver, British Columbia. Using the Ramex system, over 700 cubic meters of in-place rock were excavated from the site. The Ramex method was selected because blasting was not allowed by the local government due to adverse effects of noise and ground vibration on neighboring structures.

The Atkinson site provided an opportunity to evaluate the cutter's performance in a production environment. Unfortunately, economics and time constraints of the project did not allow for a controlled, comprehensive test program. Gathering of data for research purposes was secondary to the need to complete the job in an economical and timely manner. Only one day was allowed exclusively for testing purposes and collection of research data; however, useful production information and experience was gained that would probably not have been obtained if the site had been used purely for research purposes.

5.1 Geology

The bedrock at the site consists of pretertiary Mesozoic greenstone granite, similar to the rock at the Abbotsford quarry. The rock was unweathered and hard, averaging 11,300 psi based on Schmidt hammer tests (see Appendix C for test data). The rock was locally homogeneous except for some areas which had a higher pink orthoclase content. Bedrock was generally exposed on the surface; however, in flatter areas bedrock was covered by up to one foot of soil. Density of the rock averaged 174 pcf, as determined by three density tests.

5.2 Field Testing

Regular production excavation was suspended on June 10, 1997 to test cuttings removal systems, excavation rate and other performance characteristics of the equipment. The tests were done in an area with a relatively uniform steep slope approximately 20 feet high, that had been previously excavated by the IKC into fresh bedrock (see Figure 13). The equipment used for the testing was the same IKC, hammer and excavator

platform used at the Abbotsford quarry site. A description of the equipment is included in Appendix B.

Figure 13 Atkinson site.

5.3 Cuttings Removal Systems

Two cuttings removal methods, compressed air and compressed water, were experimented with prior to quantitative testing.

Compressed air was tested by directing a stream of high velocity air at the base of the IKC. A person standing on top of the slope directed air toward the base of the cutter through a hose with an approximately 0.5 inch nozzle on the end. To be effective, the nozzle needed to be within 1 to 2 feet of the cutter. The air was supplied by a trailer-mounted compressor powered by a diesel engine of approximately 50 hp. The compressed air appeared successful in helping remove cuttings, but was soon discontinued due to the generation of airborne dust and the proximity of the nearby residences. No quantitative data were obtained using the compressed air.

High pressure water was also tested for removal of cuttings. This method appeared successful in increasing productivity and, as an added benefit reduced the amount of airborne dust generated beneath the cutter. The water

was supplied through a garden hose from a neighboring residence, and pressurized by a standard 5 hp pressure washer. The water jet was aimed from the top of the slope down into the slot being created by the IKC. The pressurized water nozzle was held fairly close to the cutter for best results. The method was used for the quantitative tests and was incorporated throughout much of the remainder of the project. It was not used, however, while cuttings were collected for laboratory particle size analysis, to ensure cuttings were as representative as possible.

5.4 Production Testing

The purpose of the production testing was to determine the excavation rate of the IKC. Two tests were performed. For both tests the pressure washing technique, as described above, was used to facilitate cuttings removal. The production testing was done on the confined rock only. As discussed in Section 1.1, the core area between the slots can be excavated at a much faster rate due to the presence of an unconfined surface on the side of an adjacent slot, assuming the slots are spaced a reasonable distance apart. The faster excavation of the unconfined areas between the slots was not separately measured. The excavation rate was estimated to be approximately 10 times faster in these core sections than for the confined slots, based on unanimous consensus of the observers on site the day of the testing.

5.5 Production Testing Procedure

To determine slot excavation rates, first, reference points were established above and below the test area. The initial contours of the rock were established by measuring perpendicular to a straight line marked between two reference points. The operator swept the cutter down the face 10 times while the sound was recorded for later analysis. The contours of the face were then remeasured, as well as the slot width. The above procedure was repeated twice. From these data, the excavated volume could be calculated, and by analyzing the audio tape recording, the excavation rate could be determined. This procedure was repeated for a second test slot.

5.6 Production Testing Results

The first test lasted for 223 seconds. A total of 1161 cu-in. of rock was excavated for an excavation rate of 0.40 cu-yd/hr. The second test section lasted for a total of 237 seconds, with a total excavation of 891 cu-in. of rock, for an excavation rate of 0.29 cu-yd/hr. Assuming the spacing between the slots is the same as the slot widths and the unconfined core breaks out about

10 times faster, the excavation rate would be 0.73 and 0.53 cu-yd/hr, respectively, for the first and second tests.

Both the IKC and the standardmoil point tool were used throughout the project to excavate the unconfined cores. For unconfined rock the IKC appeared to break the rock at about the same rate as themoil point cutter. The optimal width of the unconfined cores was about 12 to 18 inches. If the cores were wider than 12 to 18 inches the cores did not break out as unconfined. As expected, both themoil point and the IKC broke the unconfined rock into much larger fragments than were generated from the confined core.

5.7 Cuttings size analysis

Laboratory tests were performed on rock excavated from the test face. The tested material was excavated using the Ramex cutter, without the use of compressed air or compressed water to aid in cuttings removal. The cuttings represent material from the confined test slots only, and not from the core located between the slots. The material was sieved to determine the relative particle sizes according to ASTM 425-63 (see Figure 14). This gradation of the tested material was finer than the average size from the project as a whole, since the unconfined rock from the core sections broke into much larger pieces than rock excavated from the slots. Nearly 75 percent of the excavated material was gravel sized, and the remainder classified as sand size except for less than 0.1 percent that passed the number 200 sieve. This small percentage of fines was significant as airborne dust was generated in the excavation process.

The gradation of material excavated from the unconfined cores was not tested. The average fragment size from rock excavated from the unconfined cores was significantly larger than rock excavated from the confined slots. From the sieve analysis, the largest chips produced from the cores was about 4 inches, with the average size about 1/2 inch. From the unconfined cores, the maximum fragment diameter was estimated to be about 12 inches (the distance between adjacent cores), with an average particle size visually estimated at 3 to 4 inches.

5.8 Sound Analysis

The sound generated from the vertical face slot tests was recorded with a standard portable audio tape recorder. Besides using data from the tape to

GRAVEL		SAND			CLAY OR SILT
COARSE	FINE	CRS.	MEDIUM	FINE	

Figure 14 Grain size analysis.

calculate excavation rate, the percentage of operating time and blow rate of the cutter were also determined.

The hammer frequency was analyzed with the assistance of Wizard Productions of Renton, WA. The hammer blow rate was determined by digitizing a representative sample of the recording and using commercial music production software. The printout of the digitized sound versus run time is included in Appendix D. The graph of the sound produced clearly shows a dominant wave pattern corresponding to the operation of the hammer. The results of sampling and analyzing several sections of the recording indicates an average rate of 464 impacts per minute. These test results confirmed operation within the 400 to 500 blows per minute frequency range specified by NPK, the hammer manufacturer.

From the audio tape the percentage of operating time was determined for the two test slots. This is the percentage of time the IKC is actually operating on the rock surface. The remainder of the time was associated with repositioning the cutter at the top of the slot for another pass downward. For the two tests, the hammer operated 64 and 68 percent of the time, respectively, for an average operating time of 66 percent. This percentage of average operating time was slightly greater than was achieved in Italy (Armando et al. [1994]) in production jobs using hydraulic hammers and conventional tools.

5.9 Vibration Monitoring

Macleod Geotechnical of West Vancouver performed vibration monitoring in May of 1997. The monitoring was conducted to comply with local regulations, and to determine if the Ramex system produced vibrations that could be damaging to adjacent structures. The consultant's report is included as Appendix E. The largest vibrations recorded were 8 to 10 feet away from the cutter and had a maximum value of 10.3 mm/sec (0.406 in/sec) or 0.01 gravity. The significance of these values will be discussed in section 6.5.

5.10 Atkinson Site Conclusions

Work at the Atkinson site was successful in showing that the Ramex system is a viable alternative to blasting. The project was completed on budget with the time required to complete the contract comparable to what would have been required for blasting. The project showed that Ramex's equipment can operate fairly reliably for extended periods of time. On the negative side, noise from the equipment was significant and caused some concern from the

neighbors. However, this problem has since been addressed, as Ramex developed a hammer blanket that results in significant sound damping.

6 Discussion

6.0 Ramex Method

Throughout this research the Ramex concept proved to be a viable alternative to current typical excavation methods. At this time, with the measured excavation rate of less than 1 cu-yd/hr, significant excavation could be time consuming, or would require simultaneous operation of more than one IKC. However, with continued research and development, productivity can likely be significantly improved. Excavation rates will likely be increased through development of an efficient cuttings removal system, as well as through optimization of different input energies and cutter configurations. Possible ways to increase productivity are discussed below.

6.1 Hammers

This research followed the successful adaptation of the IKC to the hydraulic hammer. The Atkinson project, being a commercial success, proved the hydraulic hammer in combination with the IKC was viable. Through many hours of operation, the hammer performed reliably and required no major repairs.

While the use of the IKC with a hydraulic hammer proved successful, the diesel ram merits further research as there may be some significant advantages over the hydraulic hammer. The diesel ram has some significant energy efficiency advantages due to the orientation of the piston in line with the cutter (refer to Section 2.3). The hydraulic hammer has energy losses in the conversion of power from the source (typically diesel), to the hydraulic pump, through the hydraulic lines, to the hammer and then into blow energy. The diesel ram has none of these energy losses. Another potential advantage of the diesel ram is the increased blow rate compared to the hydraulic hammer. Hydraulic hammers operate at a blow rate of up to 500 blows per minute for hammers in the same blow energy range as the Ramex diesel ram, which can operate at a rate of up to 700 to 800 blows per minute. This suggests that while the two hammers may have the same energy ratings, the excavation rate may be greater for the diesel ram.

The major advantage of the hydraulic hammer is its proven reliability compared to a new, largely unproved diesel ram. Another advantage of the hydraulic hammer is that only one engine is required for both the operating platform and the hammer, whereas the diesel ram system in its current setup

requires two internal combustion engines. The hydraulic hammer system could be advantageous for work in underground confined spaces, as the system could be set up so the power source, i.e., electricity, is separate from the excavator and hammer. This set-up would not require ventilation of the diesel exhaust gasses. Further, much less heat would be generated underground.

6.2 Excavation Rate

The excavation rate for a confined rock mass using the Ramex IKC tool is primarily a function of input energy, cutter design, rock type, rock hardness, rock fractures, operator skill, cuttings flushing and muck removal. Of these factors, cutter design, cuttings flushing, muck removal, and operator skill will continue to evolve and improve as more development, research, and testing is conducted. The remaining major variables, rock characteristics and input energy, can be directly compared by measuring the seismic velocities of the in situ rock, excavation rate and energy input. From this information, a data base can be developed that can be used as a predictor of excavation rates once the seismic velocities of a rock mass are determined. Development of an easy, reliable method to predict excavation rates of the equipment is critical if the IKC is to be a commercial success.

A data base of seismic velocities and IKC excavation rates can also be used to compare performance of the Ramex method to other projects where similar data are available. Such data are currently available from projects in Italy where tunneling was completed using a standard moil point tool. The use of field seismic velocities can be a relatively inexpensive method to estimate performance; however, performance cannot be estimated until a database of projects has been developed.

6.3 Energy

The grain size analysis testing as well as visual observations showed that much larger rock fragments were excavated from the unconfined cores than from the slots (see Section 5.7). The larger fragments produced from the cores corresponded to a significantly greater excavation rate, about 10 times as fast. This increased excavation rate, as well as larger fragment size, agrees with the theory [Bond 1961] that energy consumption is related to surface area of breakage, which is minimized with larger fragments. If the amount of core breakage were increased and the slot excavation were decreased, the

average fragment size would increase, resulting in a theoretical overall increase in efficiency.

In order to increase the percentage of unconfined core, the slots can be spaced farther apart. However, if the slots are spaced too far apart, the cores do not break out as unconfined. From field observations, the maximum slot spacing for the conditions at the Atkinson site was about 12 to 18 inches. This maximum slot spacing distance was for one rock type with a 3000 ft-lb/blow hammer. The spacing may be different for other rock types and conditions.

Another way to increase the percentage of core breakage is to decrease the slot width. The minimum slot width necessary to allow unconfined breakage in the adjacent cores is likely quite small due to the highly brittle nature of rock. Since the cutters used at the two test sites were of fixed width, and the minimum slot width is limited by the cutter width, excavating a core narrower than about 12 inches was impossible. The only way to excavate a narrower slot would be to build a new IKC of narrower width. A narrower cutter profile would also have the advantage of applying a higher specific energy, assuming the same hammer is used. According to the literature, (see section 2.2) a higher specific energy would produce greater efficiencies; therefore, in theory, a narrower cutter would increase efficiencies in two ways. However, a narrower cutter with the same energy input would also be subjected to more concentrated stresses, and may therefore have an increased likelihood of mechanical failure. One method to test these concepts is to machine a cutter with a narrower profile, perhaps using one less row of cutter buttons, and compare the excavation rate and reliability with the current cutter.

6.4 Cuttings Removal

Based on research at the Atkinson site, compressed air and high pressure water used for removal of cuttings resulted in an increased excavation rate. There were drawbacks to both flushing methods, however. The application of the air or water required that an extra person be present at all time when the cutter was in operation. This person was also working in a relatively dangerous position by standing very near the operating cutter. Rock chips would occasionally fly from the cutter, necessitating that the worker wear personal safety equipment. In the most recent testing, the helper was standing on the break of the slope above the operating cutter, a less than desirable location from a safety standpoint.

Energy was required for both systems with the compressed air system using considerably more energy, 50 hp vs. 5 hp. The compressed water method requires a source of clean water that may be difficult to obtain for some applications. Controlling sediment-laden runoff may be a concern for environmentally sensitive areas, even though the quantities are relatively small. Water runoff was not a problem at the Atkinson site because the waste water pooled up and drained on site into natural fractures in the rock. Controlling dust laden air would likely be a significant problem if compressed air is used for cuttings removal. This problem was significant at the Atkinson site, and no quantitative testing was performed.

Future development of the cutter design could include more research into solving the mechanical connection problems between the supply hose and the cutter head for the built-in flushing system (see Section 2.6). By incorporating the system into the cutter head, the system could be controlled by the cutter operator thus eliminating the need for a separate worker. The system could be designed to automatically operate only while the hydraulic hammer is operating, thereby minimizing the operator's duties, while reducing the water and energy consumption. A built-in cuttings flushing system might also have an added efficiency advantage by forcing cuttings away, and not into, the cutter.

6.5 Vibration

The vibrations in the rock due to the excavation process were "quite low at adjacent structures" according to the vibration consultant report from the Atkinson site. Figure 8 from Appendix E shows that the vibrations produced at a distance of 8 to 10 feet from the operating hammer were well below the limits of what is harmful to structures and machinery as well as below the limit of "troublesome to persons". The closest residence was approximately 50 feet from the nearest excavation and no vibration-related complaints were received. The peak vibrations are significantly less than if the excavation had been done by blasting, with the literature suggesting that peak ground vibrations were an order of magnitude less. However, the duration of vibrations for the Ramex method were much longer than would have been experienced with blasting. The testing overall, however, shows the Ramex excavation method would be a good alternative to blasting where peak vibrations could be detrimental to adjacent structures.

7 Conclusions

- The Ramex system appears to be an acceptable alternative to traditional rock excavation methods. Additional research, development, and testing has the potential to aid in significantly improving the productivity associated with this excavation method.
- Use of a cutter with a narrower profile to minimize less productive slot cutting and maximize the faster unconfined core breakage may significantly improve productivity. This could potentially increase efficiency in two ways: by resulting in a larger average rock fragment size (a higher percentage of rock from the unconfined core), and by increasing the specific energy which the literature suggests will lead to greater efficiency. This may be accomplished with a narrower cutter that uses fewer cutter buttons. Alternatively, use of a hammer with more energy and the same cutter configuration would increase the specific energy, although it would not necessarily maximize the amount of unconfined breakage.
- While the adaptation of the IKC to the hydraulic hammer was successful, the diesel ram may merit further study and testing in the future. The diesel ram may have inherent efficiency advantages.
- Continued development of a cuttings flushing system is critical to increased productivity. Development of a built-in system in the cutter head has potential advantages. A system that can be controlled by the hammer operator that would flush away, but not into, the cutter would likely be desirable. A pressurized water system seems to have the most promise.
- In future projects, determination of the field seismic velocity of the rock mass to be excavated would be helpful. Once a database is established for the Ramex method, more reliability will be established in estimating future projects. The data can also be used to compare the Ramex method with previous research in which measured seismic velocities were used to quantify other excavation methods.

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APPENDIX A

SPECIFICATIONS

Model	Impact Energy	Frequency	Working Weight *	Operating Pressure	Oil Flow	Tool Dia.	Tool Working Length
	(class ft-lb)	(bpm)	(lbs)	(psi)	(gpm)	(in)	(in)
H-06X	150	450~700	185	1600	2.5~4	1.6	11-1/2
H-08X	200	450~750	220	1550	4~7	1.8	12
H-1XA	300	600~800	350	1700	7~10	2.2	13
H-2XA	500	450~800	450	1850	8~15	2.6	14
H-3XA	750	500~700	700	1700	12~18	3.0	16
H-4XE	1000	600~750	1100	2200	17~22	3.5	17
H-6XA	1250	400~600	1600	2100	22~33	3.8	18
H-7X	1500	400~550	1880	2100	24~37	4.2	18
H-8XA	2000	400~550	2800	2250	26~40	4.6	20
H-10XB	3000	400~500	3300	2100	42~52	5.0	22
H-12X	4000	400~500	4400	2250	45~55	5.4	22
H-16X	5000	350~450	5650	2250	46~62	5.7	24
H-20X	6000	350~450	6600	2500	54~70	6.1	26
H-30X	10000	300~350	10600	2650	66~80	6.9	31
H-70X	20000	150~200	25500	2700	80~110	9.5	50

* Working weights may vary with bracket configuration. Specifications subject to change without notice.

Hydraulic hammer data

Table from NPK construction equipment company brochure. The 10XB was used for all testing and production excavation involved with this project. Note the range of impact energies available. Hydraulic hammers are currently manufactured by many companies.

APPENDIX B

Equipment

IKC:

The cutter used was the standard one piece unit. A standardmoil point breaker was interchanged with the Ramex cutter in areas of unconfined rock or rock previously fractured by the IKC.

Hammer:

The hammer used for the project and the testing was an NPK H-10XB with a rated impact energy of 3000 ft-lb. For all tests the hammer was charged to the manufacturers maximum recommended internal charge pressure.

Excavator:

Komatsu PC200-SL 133 hp, 45,000 pounds.

Other (used at Atkinson site only):

Honda 4-1500 GX 150 5.5 hp 1300 psi pump

Air compressor

APPENDIX C

Schmidt hammer density test data⁽¹⁾

<u>Test #</u>	<u>Reading</u>	<u>Orientation</u>	<u>Angle corr.⁽²⁾</u>	<u>Qu⁽³⁾(psi)</u>
1	32	45 deg.	34.2	10000
2	27	45 deg.	29.4	7500
3	28	horizontal	28	7000
4	43	45 deg.	44.9	16000
5	43	45 deg.	44.9	<u>16000</u>
			Average	11300

Notes: (1) values based on tested density of 174 pounds per cubic inch

(2) Angle correction based on orientation of hammer.

(3) Qu = unconfined compressive strength.

APPENDIX D

Data from audio recording of IKC and H-10XB hydraulic hammer used to determine blow rate of 464 blows per minute.

APPENDIX E

Vibration Monitoring Report

MACLEOD GEOTECHNICAL LTD.

soil mechanics and foundation engineering

SUITE 150(G), 1451 MARINE DRIVE, WEST VANCOUVER, B.C. V7T 1B8 TELEPHONE (604) 922-0812 FAX (604) 922-9167

May 29th, 1997
YH17

RAMEX ROCK EXCAVATION INC.
4043 Shone Road
North Vancouver, B.C.
V7G 2N3
Attention: Mr. Garfield C. Johnson

VIBRATION MONITORING REPORT

Vibration Source: Rock removal - "Impact Kerf Cutting"
Contractor: Ramex Rock Excavations Inc.

Dear Sir:

We have conducted vibration monitoring during initial rock excavation operations at the above noted project.

Several stations were monitored on and adjacent to the site at various distances from the work in progress. We have presented the data in graphical form for quick reference. Also attached are two reference sheets showing guidelines for vibration levels.

Noise levels were measured at the time of our monitoring and peak sound levels were in the range of 104 to 113 decibels within 50 ft. of the equipment. This monitoring was incidental to our ground vibration monitoring.

The highest recorded vibrations were measured 8 ft. to 10 ft. away from the cutting tool at 10.3 mm/s (0.406 m/s). This level is well below generally accepted thresholds for damage to structures.

As indicated by the attached data plot, the vibration level decreases over distance and therefore vibration levels should be quite low at adjacent residences.

MACLEOD GEOTECHNICAL LTD.
Glen Merkley, Technician

GLM/co

RUSSELL HOLLINGSWORTH
DESIGNER

MACLEOD GEOTECHNICAL LTD.

1451 Marine Drive
West Vancouver, B.C.
V7T 1B8

Atkinson Residence
6225 Imperial Ave.
West Vancouver, B.C.

VIBRATION MONITORING
OF
IMPACT KERF CUTTING
OPERATIONS

JOB No. YH - 17

Date: May 22, 1997

Page: 1/1

By: glm

Dwg: YH - 17 - 01

Figure 3 Potential for damage to structures combination velocity-displacement criterion (U.S. Bureau of Mines, 1980)

<p>RUSSELL HOLLINGSWORTH DESIGNER</p>	<p>MACLEOD GEOTECHNICAL LTD.</p>		<p>1451 Marine Drive West Vancouver, B.C., V7T 1S8</p>	
<p>Atkinson Residence 6225 Imperial Ave. West Vancouver, B.C.</p>	<p>REFERENCE</p>	<p>JOB No. YH - 17</p>	<p>Date: May 22, 1997</p>	
		<p>Page: 1/1</p>	<p>By: glm</p>	<p>Dwg: YH - 17 - 02</p>

GIVEN VELOCITY = 0.2 INCH/SEC.
 FREQUENCY = 10 CPS
 THEN FROM GRAPH, DISPLACEMENT = 0.003 INCHES
 ACCELERATION = 0.03g
 MOTION IS EASY NOTICEABLE OR TROUBLESOME TO PERSONS

FIGURE 8
 Allowable Amplitude of Vertical Vibrations

REF. NAVFAC DM 7.3 (1983)

RUSSELL HOLLINGSWORTH DESIGNER	MACLEOD GEOTECHNICAL LTD.			1451 Marine Drive West Vancouver, B.C. V7T 1B6	
Atkinson Residence 6225 Imperial Ave. West Vancouver, B.C.	REFERENCE	JOB No. YH - 17	Date:	May 22, 1997	
		Page: 1/1	By: glm	Dwg: YH - 17 - 02	

Primary excavation of rock with an impact tool

H. J. Handewith

Tunnel Consultant & Principal Investigator

ABSTRACT: This paper defines the principal aspects of primary and secondary rock fracture by impact. It discusses mechanical rock fragmentation and cutting tool design. It also investigates those physical rock properties that can be used to predict excavation rate performance. Specific discussions include:

- Viability of the impact-kerf-cutting (IKC) concept
- Impact rate of a tool and its effect on rock penetration within a velocity range of 10 to 20 ft/sec.
- Use of Schmidt Hammer for estimating unconfined compressive strength (UCS).
- The reliability of using UCS for production rate predictions.
- Production rate estimating procedures.

This paper covers laboratory and field testing pertaining to the development of an impact slot-cutting/ridge breaking tool for excavating rock. It was found that the IKC tool mounted on a hydraulic breaker could be competitive with explosives for trenching, tunneling, vein mining, and the excavation of other short openings in rock or concrete.

1 INTRODUCTION

Ramex Systems developed the breaker mounted impact kerfing cutter (IKC) concept for excavating, trenching and tunneling in rock. In order to determine tool design parameters and define estimating procedures for production rates, Ramex conducted a series of laboratory drop tests in two types of rock: a high strength concrete, and a low strength concrete. In addition, full scale field testing was conducted at two different quarries and at one construction site. This work was conducted under the sponsorship of the US Department of Energy's Energy Related Inventions Program (ERIP) with the cooperation of the US Bureau of Mines (now NIOSH) and the University of Washington.

The objective of this effort was to investigate the mechanics and the practicality of cutting a series of slots in a rock surface (primary breaking) and then breaking out the ridges or cores left between the slots (secondary breaking). This concept is illustrated in Figure 2. Initial field testing was conducted using a prototype diesel breaker rated at 5,500 ft-lb per blow with blow rates in the range of 440 to 480 blows per minute. The balance of the field tests were carried out using a crawler mounted hydraulic breaker with blow energy levels of 3,000 ft-lb and

blow rates of about 440 blows per minute. The initial field test cutter design is shown in Figure 1.

Figure 1. Impact Kerfing Cutter (IKC) used for the field tests

1.1 Definitions

In spite of the fact that rock excavation has its origins in antiquity, mechanized excavation is a newly evolving art. Discussing any new art requires a concise definition of the terms used. For the purpose of this report the following definitions are used:

Drilling: a method for making a hole (usually circular) in rock or concrete where a drilling fluid such as air, water, or mud is required to flush cuttings from the hole. (Waggoner 1984)

Boring: a method for making any shaped hole in rock or concrete where the cuttings are removed by gravity or other mechanical means. (Waggoner 1984)

Primary breaking: the process of initiating an opening, or a slot, in the surface of a thick and confined rock. i.e., tunneling, trenching, drilling, boring, reaming, etc. (Handewith 1990).

Secondary breaking: the process of reducing the size of a large freestanding unit of rock or concrete. i.e., slab, boulder, erratic, etc. (Handewith 1990). Cracks that are initiated on the surface usually terminate on the surface.

Figure 2. Section through a typical trench cut in rock with the IKC

Specific Energy: a measure of the energy required to fragment and reduce in size a unit volume of rock or concrete. According to Maurer (1965) "In order to break rock by mechanically induced stresses, sufficient force or energy must be applied to the rock that the induced stresses will exceed the rock's strength. Once these threshold values of force or energy are exceeded, the amount of energy required to break or remove a unit volume of rock remains nearly constant."

Table 1 illustrates the increase in specific energy when a very large sample is reduced to particles with a final size of 10-mm, 1.0, and 0.1-mm. From this it is apparent that a large portion of the energy required to fracture rock is spent generating fine and dust particles.

1.2 Single point impact and laboratory drop testing

A single point tool is incapable of excavating a deep slot in rock as each succeeding impact causes the resulting opening to shape into a cone. The sides of the cone deflect further blows without fracturing additional rock. However, a series of strategically placed single point impacts can create a linear furrow in a rock surface. Rolling button cutters that are used on raise boring machines, and continuous edge disc cutters that are used on tunnel boring machines, have long exploited this furrow or kerf cutting concept. Parallel furrows result in a breaking out of

chips between the furrows. The resulting rock chips have a maximum width equal to about 80% of the furrow spacing.

A unique feature of the IKC cutter is the two-

Table 1 Typical Specific Energy Requirements for Conventional Crushed Rock - (Maurer 1968)

step cutting process. First the cutter is used to make several parallel linear slots that are spaced such that chips freely break from between those slots. The slot cutting is *primary* breaking. To create a slot the IKC tool is translated over the rock surface by sweeping the boom of the hydraulic excavator. Repeated translations are required to deepen the slot. The resulting slot is somewhat wider than the cutter itself. The second step is to place the IKC tool on top of the free standing ridges left between the slots and break out the ridge by *secondary* breaking as shown in Figure 2.

Laboratory drop testing was conducted at the Mining Equipment Test Facility of the US Bureau of Mines in Pittsburgh in order to simulate the primary breaking action. The drop tests were conducted in 1994 and 1995 under the auspices of an ERIP grant, a cooperative effort from the University of Washington, and the US Bureau of Mines. These tests were reported by Haselton (1995) as part of his studies for a Masters of Science in Civil Engineering at the University of Washington.

One objective of the drop test was to investigate any performance difference between the high velocity-low mass impact of a hydraulic breaker and that of the Ramex diesel breaker prototype having a low velocity-high mass. A standard hydraulic breaker has an impact velocity of about 20 ft/s while that of the diesel breaker was only 10 ft/s. A second objective was to determine the effect of a blow force of 250 ft-lb per button in place of the conservative design of 125 ft-lb used on the prototype diesel breaker. A third objective was to investigate the relationships of impact energy, crater depth, and opti-

mum furrow spacing. The final objective was to develop performance estimating criteria for the IKC in several types and strengths of rock and concrete.

2.0 FULL SCALE FIELD TESTING

Field testing was conducted in three phases. The first phase was conducted at the Cadman Rock Quarry situated near the city of Monroe Washington. A 5,000 ft-lb low-velocity prototype diesel impact breaker developed by Ramex was used. The last two field tests used a standard 3,000 ft-lb crawler mounted hydraulic breaker. It was used in a rock quarry situated near Abbotsford British Columbia, and then for a granite foundation excavation in North Vancouver British Columbia.

2.1 *Operation of the IKC cutter*

The machine operator determines the optimum slot depth and spacing between slots for each type of rock through trial and error (see Figure 2). The ridges between the slots are broken out by secondary breaking. The advantage of impact Kerf Cutting is realized when one looks into the energy required to break out the free standing ridges left between the slots. This ridge or "core breaking" operation produces very large rock cuttings, often in the range of minus ten inches. Quite simply, the larger rock fragments produced by ridge breaking (secondary breaking) consume disproportionally less energy than what would be required to reduce the rock to smaller fragments. Ridge breaking results in larger cuttings and a more energy efficient rock fracturing process.

2.2 *Cadman Rock Quarry and testing with the prototype diesel breaker*

For initial testing the prototype IKC cutter was mounted on a prototype diesel breaker at the Cadman High Rock quarry situated near the City of Monroe Washington (Handewith 1989). This quarry is noted for producing durable riprap, landscaping, and building stone. The quarry rock is a hard and abrasive basaltic-andesite. It has an unconfined compressive strength ranging from 25,000 to 40,000-psi. In some areas the rock was fractured due to prior blasting. In general the rock had a close set, moderately tight joint system spaced at 4 to 8 inches.

The original IKC cutter was conservatively designed for an average force load of 125 ft-lb on each tungsten carbide button. It was designed to last throughout the development operation. Maximum working life was the primary consideration with manufacturing cost and rock cutting efficiency secondary considerations. The diesel breaker had a total of about 100 operating hours when operations were shut down in early 1991. About 50% of the operating time could be considered 'full-load testing' (i.e., 5,000 ft-lb per blow). Two cutters were used over the prototype test period. The first IKC failed when one of its 52 tungsten carbide buttons sheared off due to improper handling. This occurred after about 30 hours of operation. Though the cutter remained functional, it was likely that the missing button could influence the cutters performance studies. It was removed and used as a spare. The second cutter completed the additional 65 hours of testing, showing no sign of wear.

Once the load, blow rate, and production rate capabilities of the diesel breaker system were verified an air/water-mist sub-system was tested to flush cuttings from under the cutter. This was done to minimize inefficient regrinding of the rock cuttings under the cutter. About two gallons a minute of water were injected into the 120-psi (at 250 scfm) air system. This system proved effective at removing rock cuttings from under the cutter, and reduced the occurrence of airborne dust. However, airborne respirable dust remained a problem. As reported by Kovseck, et. al., in 1991, the mitigation of airborne dust needs more work. This is a problem with all hydraulic breaker operations and is not specific to use of the IKC cutter. Kovseck, et. al., also found that noise, ground vibration, and dust generation levels were nearly equivalent to that of a standard hydraulic breaker with the same blow energy working in like rock conditions.

2.3 *Abbotsford Quarry Testing Using a 3,000 ft-lb Hydraulic Breaker*

Full-scale testing was conducted at the Abbotsford quarry in February 1997. The quarry is situated in a massive greenstone outcrop just outside the city of Abbotsford in British Columbia. The testing was conducted to demonstrate operation of the pre-production IKC cutter. It was reported by Capell (1998) as part of his studies for a Masters of Science in Civil Engineering at the University of Washington. This new cutter was 9" wide by 16.5" long and supported 27 face and 4 gage tungsten carbide buttons. (The original cutter was 16 inches

square and with 52 tungsten carbide buttons.) The hydraulic breaker was an NKP H-10XB with blow rating of 3,000 ft-lb. The cutter button force-load was calculated to be the same as observed at Cadman in the range of 110 to 125 ft-lb per button. The breaker was mounted on a Komatsu PC200-SL 45,000-pound hydraulic excavator. (Figure 3).

Figure 3. Sketch of trenching with the IKC mounted on a hydraulic breaker

Testing was conducted on an undisturbed outcrop of greenstone with an exposed surface angle about 30° above vertical. The angle of the slot was greater than the angle of repose allowing the cuttings to fall freely from the slot by gravity. A series of 9-inch deep slots was cut and measured for repeatability. It was found that the breaker was operating at a rate of 464 blows per minute and mining rock at an average rate of 2.4 cubic feet per minute. This calculated to an average IKC sump rate of 0.06 inches per blow and a production rate of 5.3 cubic yards (13.3 tons) per hour.

The pre-production IKC was supplied in two pieces. The flat plate cutter, mounting 27 tungsten carbide buttons, was bolted to an upper shaft. Periodic loosening and breaking of the bolts attaching the cutter plate was a problem. It was elected to re-design using a single unit cutter and shaft for future testing. Other than the bolts, the cutter showed no wear after eight hours of operation over the two day period. These results were significantly promising to encourage further testing.

2.4 Foundation Excavation and Production, Testing with a 3,000 ft-lb Hydraulic Breaker

Ramex contracted to excavate a 700 cubic meter granite foundation in the wealthy suburban area of West Vancouver British Columbia. The city council had prohibited the use of explosives and the only alternative available was some form of mechanical excavation. The property is situated in a coarse grained granite of the West Vancouver pluton. The results of testing at this location is reported

by Capell (1998). This project employed the same 3,000 ft-lb hydraulic breaker used at the Abbotsford quarry. It had an average cutter button force load of about 125 ft-lb per blow.

The estimated IKC mining sump rate was on the order of 0.09 inches per blow (calculating to an average mining rate of 3.6 insitu cubic yards pre minute). Working a full 60 minute hour would provide a production rate of 8.0 cubic yards per hour (17.5 tons/hour). Based on the laboratory drop tests it is expected that these production rates would be doubled to 16 cubic yards per hour using a 6,000 ft-lb hydraulic breaker.

3 RESULTS AND TEST DATA

Considerable data were collected from the laboratory drop tests, the two quarry tests, and the one production run. These data are reported in the reference material and this report is limited to those parameters that influence production rate and production rate estimating.

3.1 Statistical investigations of test data

Data from the laboratory drop tests shown in Table 2 and the full-scale field testing in Table 3 were combined for the statistical linear regression analyses shown in Tables 4 and 5.

D'Andhrea et.al., (1965) reported that compressive strength could be predicted from other known rock properties. Several other investigators have reported a statistically significant relationship between the Schmidt Hammer rebound value R and the unconfined compressive strength of rock. Table 4 affirms those early investigations. Aufmuth (1972) reported that the Schmidt Hammer rebound R value has a reasonable correlative linear regression relationship of ($r^2 = 0.54$) with unconfined compressive strength. This relationship was enhanced ($R^2 = 0.92$) when the R value was multiplied density of the rock, in pounds per cubic foot (pcf) as reported by Miller in 1965.

The rock strength parameters in Table 2 and Table 3 were combined and statistically compared, using linear regression methods, for correlation with IKC penetration in each type of rock and concrete. The results are shown in Table 4 and Table 5. This study revealed that Young's Modulus (static) was a strong indicator of IKC rock penetration and that UCS was a reasonable indicator of IKC penetration. Contrary to expectations, there was no improvement in penetration correlation when the UCS was divided by the bulk density (pcf). The Schmidt Hammer R value by itself indicated little correlation with IKC

penetration ($r^2 = 0.31$). However, when the R value was modified the bulk density ($R \times \text{pcf}$) the correlation coefficient improved ($r^2 = 0.66$). This indicated that the Schmidt Hammer can be used as field tool to provide a reasonable estimate of IKC penetration for IKC production rate estimating.

Table 2 Laboratory drop test results

	Specimen			
	A	B	C	D
UCS (psi)	3620	9915	6700	28600
Tensile Strength (psi)	360	572	155	1110
Young's Modulus (psi x 10^6)	3	5	2	6.6
Density (pcf)	135	142	132	165
Schmidt Rebound R	39	52	52	70
IKC Penetration (in/blow) at 250 ft-lb	0.33	0.15	0.34	0.08
IKC Penetration (in/blow) at 125 ft-lb	0.16	0.07	0.17	0.04

A = Low strength concrete

B = High strength concrete

C = Berea sandstone

D = Barre granite

A cautionary note is required on the operation of the Schmidt Hammer. Because this device is very sensitive to the operator it must be used according to the procedures outlined by Miller et.al., (1965)

3.2 Production rates and estimating

The laboratory and field-test data indicated that when the rock UCS or the Schmidt Hammer R value and bulk density (pcf) are known, a reasonable production prediction could be made. Even though conditions were controlled for this report, it must be cautioned that production with a hydraulic breaker is governed to great extent by equipment condition and operator judgement.

Table 3 Full scale field testing

	Specimen		
	A	B	C
UCS (psi)	32000	21500	11300
Tensile Strength (psi)	2600	1400	790
Young's Modulus (psi x 10^6)	7.5	6	5.8
Density (pcf)	178	186	174
Schmidt Rebound R	62	46	37
IKC Penetration (in/blow) at 250 ft-lb	0.08	0.12	0.18
IKC Penetration (in/blow) at 125 ft-lb	0.04	0.06	0.09

A = Basaltic Andesite

B = Greenstone

C = West Vancouver Granite

Table 4 Schmidt Hammer (R) vs. Unconfined compressive strength (UCS)

	r^2
R as an indicator of UCS	0.54
($R \times \text{pcf}$) as in indicator of UCS	0.92

r^2 = Linear regression best fit

pcf = density (pounds per cubic foot)

Table 5 Statistical indicators of IKC penetration

	r^2
Young's Modulus vs. penetration	0.92
UCS vs. penetration	0.75
(UCS X pcf) vs. penetration	0.74
Schmidt Hammer ($R \times \text{pcf}$) vs. penetration	0.66
Tensile Strength vs. penetration	0.58
Schmidt Hammer R vs. penetration	0.31

r^2 = Linear regression best fit

pcf = density (pounds per cubic foot)

Slot excavation and ridge breaking are two distinct production operations. Table 6 lists the production rate estimates for slot cutting based on the observed field data and Table 7 lists the combined slot-cutting/ridge-breaking production estimates

Table 6 Production rate estimates for slot cutting (Cubic yards per hr)

Specimen	Breaker Rating	
	6,000 ft-lb 400 bpm	3000 ft-lb 460 bpm
Low UCS Concrete	25.8	14.4
High UCS Concrete	11.7	6.3
Berea Sandstone	26.6	15.3
Barre Granite	6.3	3.6
Basaltic Andesite	6.3	3.6
Greenstone	9.4	5.4
Vancouver Granite	14.5	8.1

*Blows per minute

Table 7 Production rate estimates for combined slot cutting and ridge breaking (yd³/hr)

Specimen	Breaker Rating	
	6,000ft-lb 400 bpm	3,000 ft-lb 460 bpm
Low UCS Concrete	36.1	20.2
High UCS Concrete	16.4	8.8
Berea Sandstone	37.2	21.4
Barre Granite	8.8	5.0
Basaltic Andesite	8.8	5.0
Greenstone	13.1	7.1
Vancouver Granite	19.7	11.3

*Blows per minute

From the production results in Table 7 and Table 8 it is clear that slot-cutting/ridge-breaking production rates up to 36 cubic yards per hour can be achieved using the Ramex IKC cutting tool.

4 CONCLUSIONS

1. Impact-kerf-cutting (IKC) is a viable rock excavation concept. It can be production rate competitive with explosives.
2. Tool impact velocity had little if any effect on rock penetration within the velocity limits of 10 to 20 ft/sec tested for this study. The IKC performed equally on the diesel breaker and on the hydraulic breaker with the same button loads.

3. A high velocity air/water mist system is necessary to clean cuttings from under the cutter and to mitigate the generation of airborne respirable dust.
4. The Schmidt Hammer can provide a reasonable estimate of UCS. And when the hammer data is modified by the bulk density of rock this becomes a highly reliable indicator of UCS.
5. Production rates can be reasonably estimated.
6. Both Young's Modulus and UCS are reliable rock properties to use for estimating IKC production rates.

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Recent Attempts to Supplement Explosives with a Mechanical Excavation System

By Howard J. Handewith
16 November, 2000

The Mechanical Pre-Cutting Tunneling Method (MPTM)

by E. (Walt van Walsum, Pointe Claire, Quebec, Canada

Proceedings 1993 RETC p 129-145

This method is only practical in rocks up to 80 MPa UCS

Self-Propelled Abrasive Jet Reamer/Miner for Deviated Boreholes in Hard Rock

John H. Archibald, Arthur L. Miller, and George A. Savanick

US BOM Twin Cities Research Center, Minneapolis, NM

Proceedings 1991 RETC p 829-839

This is a 'down-the-hole' selective mining device. Patented - features include:

Propel itself in and out of an 8-inch borehole

can cut hard rock with the ab-jet and transport cuttings to the surface

can negotiate deviated boreholes - it can follow a vein and extract only the values.

A Rock Excavation system Based on Penetrating Cone Fracture Technology (PFC)

Brian P. Micke, John D. Wason, Chapman Young III

Proceedings North American Tunneling '94 - AUA/CSM p 2B19 to 2B-34

Sunburst Excavation, Inc., 303 East 17th Ave., Suite 700, Denver CO 80203

Rock is broken efficiently and with relatively benign flyrock by a system based on the use of commercial propellants. Some of the performance characteristics, such as powder factors and yields per shot, are presented along with environmental characteristics, such as seismic, dust, noxious fumes and noise signatures. A continuous excavation machine based on PCF technology will be trailed in Australia beginning in June 1994. The rationale behind the design of this machine and the manner in which the technology is being introduced to the mining and civil construction industries is discussed.

Rock Splitting as a Primary Excavation Technique

J. Paraszczak & J. Hadjigeorgiou

Dept. of Mining & Metallurgy, U of Laval, Quebec City, Canada, G1K 7P4

Proceedings North American Tunneling '94 - AUA/CSM p 2B-34 to 2B47

Discusses the use of rock-splitters in underground mining operations.

Mini-Cutter Technology - The Answer to a Truly Mobile Excavator

J. E. Friant, L. Ozdemir & Erika Ronnkvist EEA & CSM

Proceedings North American Tunneling '94 - AUA/CSM p 2B-34 to 2B47

Discusses successful laboratory tests - few field test results

Performance and Experimental Development of the Mobile Miner at Mount Isa

Excavating Tools for Rock and Soil

GROUND PENETRATING MEANS & METHODS

	BORING (mechanical cuttings removal)				DRILLING (fluid cuttings removal)			SAWING & FRICTION (friction) (soil/earth)	
	Rolling Disc (kerfing)	Rolling Cone	Impact	Drag (kerfing)	Rolling Cone	Impact	Drag (kerfing)	Sawing	Displacement
Average Chip Size	0.5"	0.5'	>>0.5"	>0.5"	<<0.5"	<<0.5"	<<0.5"	<<<0.5"	N Ap
Prime Cutting Force	Thrust	Thrust	Thrust	Torque	Equal	Equal	Equal	Torque	Thrust
Specific Cutting Energy	Moderate	High	Low	Moderate	High	Moderate	High	Very High	Very Low
CUTTING TOOL	Disc	Cone	Moi Point or Spade	Pick or Point Attack	Cone	Bit	Pick	Saw	Pipe or Shield Edge
* cutting edge	Steel or TC	TC or Steel	Steel	Steel, TC or PCD	Steel or TC	TC or Steel	Steel, TC or PCD		steel edge ripper tooth
* rock contact footprint	Line	Button	Point or Line	Point	Button	Point	Point	Blunt Edge	Sharp Edge
* Tool induced stress	Moderate	High	Very High	Low	High	Very High	Low	Moderate	High
* tool footprint area	large	moderate	low	low	moderate	low	low	low	moderate
MACHINE TYPE	Tunnel Boring Machine	Raise Boring Machine	Hydraulic Hammer	Roadheader Continuous- Miner	Blast hole Drill	In-the-hole hammer	Rotary Drill	Diamond Saw Core Drill Wire Saw Water-jet Abrasive-jet	Tunnel Shield

GROUND CUTTING RANGE

ROCK STRENGTH (1)	BORING (mechanical muck removal)				DRILLING (fluid muck removal)			SAWING & FRICTION (friction) (soil/earth)	
	Rolling Disc	Rolling Cone	Impact	Drag	Rolling Cone	Impact	Drag	Sawing	Displacement
Very High over 32,000 psi		↑	↑		↑	↑		↑	
High 16,000 to 31,000 psi	↑				↑			↑	
Medium 8,000 to 16,000 psi		↑				↑			
Low 4,000 to 8,000			↑	↑			↑		
Very Low Less than 4,000 psi	↑							↑	↑

(1) Deere & Miller, 1966

TC = Tungsten Carbide
PCD = Poly-crystalline Diamond
N Ap = not applicable

< less than
> greater than

Paper to be presented to the ISDT, May 1990

RAMEX Mining System - Update

by

Howard J. Handewith, Marketing Manager

William D. Coski, Chief Engineer

RAMEX Systems Inc.

Bellevue WA

I INTRODUCTION

To follow-up on a question asked by president Don Zeni at the last ISDT Annual Meeting regarding the extent of Ramex research, and to stay in tune with the topic of the current Annual Meeting, which is the underground construction and mining markets, RAMEX Systems has identified a world-wide market potential of \$400 to \$500 million for mining of shafts and tunnels in medium to very high strength rock. This market is now being served by explosives. Due to the perceived danger of explosives in hazardous and urban environments, 10 to 15% of this marketplace is demanding an alternative mining method that could be cost competitive with explosives. Boring machines for tunnels, raises and shafts have a limited, but, clearly established beach-head on a portion of this market. Roadheaders and continuous miners only slightly affect medium to hard rock excavation and mining operations. Hydraulic impact breakers have come into this arena, with a fair degree of success, for mining soft to medium strength rock. The need then, is for a light-weight highly mobile mining machine that can extend any shape opening in rock classed as medium to very high strength. Such a machine must be production rate and cost competitive with the use of explosives.

The patented* RAMEX mining system appears to fill this market need. Now, to get back to Don's question of why so much research? The value of the total population of hydraulic breakers in the world today is estimated at about \$750 million. Two large companies developed the original hydraulic breakers. Neither of these two international corporations now produce hydraulic breakers. The apparent reason is that they did not follow-through with their research and development programs. These two companies did all the basic research and left it to others to complete development and profit from this lucrative market. I think it was Disraeli who said "*those who do not study history are condemned to repeat it.*" In short, this is the best reason I can give for the extensive research program RAMEX Systems has undertaken in cooperation with the U.S. Bureau of Mines.

An expected spin-off of this research is to provide the mining and construction industries with a useful qualitative method of estimating rock strength properties that are related to mechanical mining systems. We all know that unconfined compressive strength (UCS) alone is not the answer, but, for now it is the only universally recognized measure of rock strength that we have. For the last 4,000 years the construction industry has relied on some form of UCS to determine safe ground foundation properties and building stone strength. The pyramids were probably designed by stacking blocks until the one on the bottom failed. We do essentially the same thing today, only now we have the convenience of a carefully controlled hydraulic press and exact dimensional control of the unconfined rock or concrete sample. According to *Engineering News Record*, (1/18/90) UCS testing lacks confidence for building stone or concrete design. For buildings, the test results should be exact or low to provide safe design criteria. For mechanical mining rate predictions we want the rock to fail with each load cycle. Mining rate predictions, based on UCS, would be useful only if the test results are accurate or high. Is it reasonable to correlate UCS with mechanical fracturing of a confined rock that is infinitely thick, and where failure is required with every load cycle? This leap in faith has led to many construction disputes, in the last 20 years, since mechanical boring machines have become a standard construction and mining tool. We need some other value system to determine the resistance of rock to mechanical mining methods. One facet of our research is to take a close look at this area using modern analytical methods.

At this point I would like to discuss the definitions in Table 1. We are the Institute of Shaft Drilling Technology not Shaft Boring Technology. Using these definitions it is easy to see the difference between "drilling" and "boring". As an example; a raise boring machine or raise drill, "drills" down and "bores" back up for the reaming operation. It was good to see just how well these definitions fit the equipment, cutting tools, and rock breaking specific energy requirements. The difference between "excavating" a partial face and "boring" a full face is another one worth thinking about. Maybe the ISDT should think about establishing a Standards Committee to assure that we are all referring to the same operations.

Why be concerned with specific energy? Power is not expensive and the cost advantage of faster construction will usually offset any increase in power costs. However, energy not used in breaking rock must go someplace. It goes into flexing hydraulic system components, wear on equipment components and wear on cutting tools. This all leads to increased equipment operating expense and production down-time. Efficient delivery of energy to the rock will reduce operating costs, equipment down-time and lead to faster and less expensive progress rates. It is clear that larger rock fragments require less energy to produce. If you can deliver the same horsepower level to the rock and produce larger cuttings you will get a reduced specific energy and faster progress rates. The RAMEX cutting principle (patent applied for) incorporates the combined advantages of impact, kerf slot cutting, and rock core breaking.

To evaluate performance of the RAMEX prototype, we are using specific energy as was originally suggested by Jim Friant from Robbins (1980). Ozdemir (1989) used both values of Hp-Hr/ton and Hp-Hr/Yd³. D'Andrea et al. (1965) and Judd and Huber (1962) suggested a possible relationship between rock density and unconfined compressive strength and other rock properties. RAMEX is using the Hp-Hr/ton value as it makes some provision for rock strength. All mechanical rock fracture is initiated by the two basic mechanisms of elastic-compressive deformation and tensile and/or shear crack propagation. The specific energy required for mechanical rock breaking can vary from a low of 0.14 horsepower-hours per ton with an impact breaker, to a high of 400 horsepower-hours per ton, or more, for deep oil drilling operations.

Table 2 provides an over view of mechanical rock penetration methods now in use. Even with explosives we must first drill a hole. The ideal cutting tool should meet the following criteria.

- Economically mine any rock ranging from "very low" to "very high" strength.
- Produce large size rock cuttings. (not possible in drilling operations)
- Require the lowest possible specific energy
- Require thrust as the principle force.

The search for an optimum cutting tool led from an evaluation of the tools in use listed in Table 1, to possible combinations of systems and materials such as water-jets to assist or cut fine slots in rock and polycrystalline diamonds. It was concluded that a cutter needed to "kerf-cut" slots in rock producing large cuttings with a low specific energy. An ability to breakout cores between the slots could produce larger cuttings and reduce the specific energy even more. The tool should require thrust as a principle force to reduce drive system complexity and provide maximum operational reliability. This search lead to development of the impact slot-kerfing cutter.

IMPACT SLOT-KERFING CUTTER (patent pending)

The 16 inch square impact kerf-slotting cutter is bolted to the forward end of the ram. The prototype cutter, Figure 1 , carves a series of equally spaced slots about 16 inches wide and several inches deep. Moving the boom over the tunnel face will produce a deep slot as shown in Figure 2. Early research indicates that this impact kerf-slotting operation requires less specific energy than a tunnel boring machine and it produces rock cuttings of about the same size. The ram cutting action comes into its own when it is used to break-out the large rock cores left between the slots as shown in Figures 3 and 4. These cores, being unconfined, break freely to both sides. This cutting action then produces large rock chips and provides a considerable reduction in specific rock breaking energy.

THE PROTOTYPE RAM

In early 1989 RAMEX Systems, Inc. of Bellevue, Washington, rolled-out the experimental prototype impact mining ram. This ram encloses a 700 pound free-floating piston which is driven by a two stroke single cylinder diesel engine shown schematically in Figure 5. This simple engine is capable of developing about 130 horsepower and delivers 100 horsepower to the rock. It is capable of delivering 5,500 ft-lbs of blow energy up to 700 times a minute.

To start, the piston is driven into the combustion chamber by an external compressed air system. A linear cam shaft attached to the piston drives a pump that sends fuel through the injector just before the piston reaches the top of the combustion chamber. Heat generated ahead of the piston ignites the fuel. The piston is driven by diesel combustion back to the bounce chamber at the cutter head end of the ram. Air in the bounce chamber, acting as a spring, is compressed as the piston advances and arrests forward motion, stopping the piston before it hits the bottom of the bounce chamber. This transfers the piston energy to the ram and hurls the ram and cutter into the rock. Energy stored in the compressed air spring forces the piston back toward the combustion chamber for the next diesel stroke. There is no metal-to-metal contact.

In practice, the RAMEX Ram must be pre-loaded into the rock. A thrust force of 15,000 pounds is required to sustain a uniform excavation operation. Using a crawler to ground coefficient of 0.3 would require an undercarriage with a weight of just over 25 tons for the 100 horsepower prototype ram. For shaft drilling this would be a requirement for the dead weight of the galloway. Grippers or stelling jacks are unnecessary.

TEST PROGRAM & Cooperative Research

The joint cooperative research included RAMEX Systems of Bellevue Washington, the U.S. Bureau of Mines, and Schneider Services of the Pittsburgh Research Center. General objectives were:

- A. Evaluate performance of prototype mining machine while working under normal mining conditions.
- B. Establish base line operating conditions.
- C. Determine efficiency of energy transfer from:
 - 1. the floating piston to the rock cutter
 - 2. the rock cutter to the rock

The following parameters were monitored:

- A. Piston and rock cutter location
- B. Diesel combustion pressure
- C. Fuel injection pressure
- D. Ram/rock cutter acceleration
- E. Triaxial force reactions on the ram undercarriage
- F. Ground motion on the rock face and the quarry floor
- G. Dust make:
 - 1. Near the rock cutter
 - 2. Near the machine operator

The Bureau of Mines sponsored Schneider Services to make three joint data collection visits to the project. The data are now being reduced at the Bureau of Mines in Pittsburgh and we expect to publish the findings sometime in 1991.

Preliminary Production Estimates:

The production rate, Figure 6 , based on the estimated volume of rock removed and the average displacement of the cutter into rock, was just under three (3) horsepower-hours per ton. Rock is a 40,000 psi (UCS) felsitic andesite, which is also very abrasive. Based on the estimated production of three horsepower-hours per ton and 100 ram horsepower to the rock, the expected mining rate is about 33-1/3 tons per hour. In a square section, 12 by 12 foot tunnel this mining rate suggests an penetration rate of 2.7 feet per hour. Allowing 50% of the shift working time for positioning the boom, ground support, muck disposal. etc., one could anticipated a progress rate of 10.8 feet in an eight hour shift for an overall rate of 32.4 feet in a three shift day.

* U.S. Patent Number 4,332,420

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TABLE 1 - DEFINITIONS

BREAKING: Breaking of a free standing boulder or rock sample with 40 to 50% of input power (mechanical efficiency) delivered to the rock.

Equipment: impact breakers, hydraulic wedges, chemical swelling

Cutting Tool: moil point, spade, wedge

Cutting Action: mostly impact with normal force (Fn)

Specific Energy Range: 0.14 to 0.28 Hp-Hr/ton

EXCAVATING: Partial face - ripping, trenching, or dredging an exposed rock face with 50% of input power delivered to the rock.

Equipment: continuous miners, roadheaders, rippers, excavators, tunnel shields, impact breakers, etc.,

Cutting Tool: pick, ripper tooth,

Cutting Action: drag and/or rotary, high cutting force (Ft) low normal force (Fn)

Specific Energy Range: 1.42 to 3.17 Hp-Hr/ton

BORING: Mechanical removal of cuttings (NAS USNCT/T, 1984) full face, usually kerf cutting with 50 to 60% of input power to rock.

Equipment: tunnel, raise and shaft boring machines and box hole drills.

Cutting Tool: rolling disc, button or tooth, and drag pick

Cutting Action: rotary

rolling cutter, with high normal force (Fn) and moderate cutting force (Ft)

drag cutter, with high cutting force (Ft) and moderate normal force (Fn)

Specific Energy Range: 4.48 to 18.07 Hp-Hr/ton

DRILLING: Any drilling operation requiring the fluid removal of cuttings. (NAS USNCT/T, 1984)

ROCK: Holes of short length (usually less than 100 ft) made in any direction with less than 35% of input power to rock.

Equipment: rock drills, blast hole drills, etc.,

Cutting Tool: drag and/or rolling disc, button or tooth

Cutting Action: rotary and/or impact

Specific Energy Range: 100 to 300 Hp-Hr/ton of drilled hole.

OIL WELL: Very long holes (usually thousands of feet) of vertical or nearly vertical holes drilled from the surface, with less than 25% of input power to rock.

Equipment: drill rig

Cutting Tool: mostly button bit

Cutting Action: rotary, though impact is used in rare cases

Specific Energy Range: 200 to 400 Hp-Hr/ton of drilled hole.

PRINCIPAL FORCE

The governing force reaction on a cutting tool, either torque or thrust; drag picks normally require more torque than they do thrust. Disc cutters require a torque force of 15 to 25% of the applied thrust force, and impact cutting tools are governed totally by thrust.

PENETRATION METHOD	BORING				DRILLING			CUTTING - (friction)	
FORCE MODE	Rolling Disc (kerfing)	Rolling Cone	Impact	Drag (kerfing)	Rolling	Impact	Drag (kerfing)	Sawing	Soil or Earth Displacement
Average Chip Size	> 0.5"	0.5"	>> 0.5"	> 0.5"	<< 0.5"	<< 0.5"	<< 0.5"	<<< 0.5"	n ap
Principal Force	Thrust	Thrust	Thrust	Torque	Thrust	Thrust	Thrust	Torque	Thrust
Specific Energy	Moderate	High	Very Low	Low	High	Moderate	High	Very High	Low
Cutting Tool	Disc	Cone	RAMEX (1) Moi Point Spade	Pick Point Attack	Cone	Bit	Pick	Diamond Blade Core Drill Wire Saw Water-jet	Pipe or Shield Edge and/or Ripper Tooth
Rock Contact Material	Steel	T.C. Steel	Steel T.C.	T.C. Steel PCD	T.C. Steel	T.C. Steel	T.C. Steel PCD	Diamond or other Abrasive	Steel
Rock Contact Geometry	Line	Button Tooth	Point Line Button	Point U Edge	Button Tooth	Button Tooth	Point U Edge	Blunt Edge	Sharp Edge

Rock Strength (2)

ECONOMIC CUTTING RANGE

Very High over 32,000 psi		↑	↑		↑	↑		↑	
High 16,000-32,000 psi	↑								
Low 8,000-16,000				↑			↑		
Very Low Less than 4,000 psi									↑

NOTES: 1. Patent Pending

2. Deere & Miller, 1966

T. C. = Tungsten Carbide

PCD = Poly-crystalline Diamond

< = less than

> = greater than

n ap = not applicable

TABLE 2

FIGURE 1
RAM MOUNTED ON HYDRAULIC EXCAVATOR
 Showing Impact slot-kerfing cutter on right
 end of ram.

FIGURE 2
WIDE SLOT CUT IN QUARRY FACE
 showing grooves cut by the slot-kerfing cutter
 action. Kerf spacing is 3-1/2 inches.

FIGURE 3
ROCK CORES LEFT BETWEEN SLOTS ON TUNNEL FACE

FIGURE 4
ROCK CORES IN 40,000 PSI (UCS) ROCK
Shown just before break-out test

FIGURE 5
SCHEMATIC REPRESENTATION OF DIESEL DRIVEN RAM

FIGURE 6
SPECIFIC ENERGY vs MINING RATE
Estimated for 100 horsepower RAM with 5,500 ft-lb blow at 700 blows per minute. This curve is for cutting the slot. Core breaking will have a lower specific energy allowing faster mining rates.

DEVELOPMENT OF A NEW IMPACT MINING SYSTEM

by Howard J. Handewith

Talk to be presented to the ISDT Annual Meeting in Las Vegas,
April 19 to 21, 1989

The concept of using impact to create underground openings is not new. Dr. Lynn Willies (Ref 1) in describing the working of the 2,300 year old lead-zinc-silver mines in Rajasthan, stated that "Pillars seem to have been worked by hollowing-out using fire, with the thin wall remaining possibility removed using a 'battering-ram': two wood bipods suitable for suspending a ram were found adjacent to one such pillar which has mechanically-induced damage." 2,300 year old "mechanically-induced damage" made by a battering-ram for mining rock is impressive. More recently and more practically, however, work conducted by Beus and Phillips in 1981 (Ref 2) indicated the practical application of a 36.4 horsepower pneumatic-powered impact rock breaker for shaft sinking operations.

Shaft sinking technology is slow in evolving, partly because of the conservative nature of the industry and partly because of the need to sink shafts in a proven way to satisfy contractual and profit needs. Capital risks cannot normally justify the use of a wholly new or revolutionary excavation system. Yet the need exists to improve our ability to sink shafts in a safer, faster and more economical way.

Several attempts to implement revolutionary shaft sinking systems in the past have been less than successful, partly, because the financial commitment for follow through could not be sustained. And yet some of these attempts are, even yet, viable. On the other hand, slowly evolving concepts such as blind boring for mechanical shaft sinking have progressed and are gradually gaining acceptance in the industry. The inflexibility of most mechanical boring systems such as a fixed and circular diameter have also limited industry acceptance of mechanical shaft sinking technology.

Today in Italy heavy-duty hydraulic breakers are being used to construct several major tunnels in soft sedimentary rock. One contractor Cogefar, advanced the Capo Nero rail tunnel, at a rate of 33 feet each 24 hour working day. (Smith, 1988) Breakers mounting amoil point tool worked well in the soft and somewhat fractured rock. The Italian contractors have clearly demonstrated safety, economy and efficiency using impact mining on a soft rock tunnel face. Featuring the flexibility of the more familiar drill jumbo, the relatively low initial cost of the mounted breaker allowed the use of two or more units on each project. The problems of mining hard rock with impact, though still present, are dampened in the softer sedimentary formations.

Lets consider definitions for a minute, so that we don't confuse apples and oranges in the rest of this talk. Horsepower is a good measure of an impact breaker, because it accounts for both ft-lbs per blow and blows per minute and it provides a simple measure of the energy transferred into rock. This leads us to an even more interesting term and that is horsepower-hours per ton. This specific energy term gives us the ability to compare any mechanical

mining method with a given rock type. As an example the 36 horsepower breaker, reported by Beus and Phillips, mined 2,472 cubic yards of a 28,000 psi (ucs) quartz monzonite. It did this in 119.6 hours of operation. If the rock has an in situ weight of 2.3 tons per cubic yard the energy/breakage rate is 1.3 horsepower-hours per ton. "Horsepower-hours per ton" though not an exact measure, at least provides us with a quantitative "flavor" of a given mechanical mining method and its rock breaking interaction. (see Table 1)

Energy costs are a small percentage of any shaft or tunneling operation. A larger power cable or a few more kilowatts should have only a small influence on total excavation costs. However, higher energy requires a larger initial cost of equipment and the lost power must go someplace. It is usually manifest as heat, friction, wear and other such items that result in lost time and high maintenance costs. Lower rock breaking energy rates should result in faster advances and a lower costs.

The observation in Table 1 that impact breaking was so much more efficient, stimulated the research that led to the development of the RAMEX rock excavation system and to the purpose of this talk. The RAMEX Impact Mining System is offered for consideration as a new development in the continuing evolution of tunneling and shaft sinking technology.

THE RAMEX TUNNELER

The modern hydraulic breaker (Figure 1) has evolved from the pneumatic jack hammer. The result is a small mass with a high velocity impact that is nearly as capable of damaging itself and the mounting equipment as it is of damaging the rock. The dynamic hydraulic drive system losses require that the input horsepower be two times the energy delivered to the rock.

Employing the same basic principal as they did 2,300 years ago in Rajasthan, the 100 horsepower prototype RAMEX Mining Boom hurls its 3,500 pounds of mass into the rock. Unlike the old days, this is now done at a rate of 700 blows each minute. The result is a low velocity, high energy impact of 5,500 ft-lbs force. Something over 115 horsepower is generated in the ram diesel combustion chamber; a 700 pound piston is hurled forward and 100 horsepower is delivered via the ram through the cutter into the rock. (Figure 2) The piston, also internal to the ram, is caught at the bottom stroke in a pneumatic bounce chamber, which compresses the air and returns the piston for the next diesel cycle. The blow energy reaction is taken out of the system by the piston mass during the return stroke and very little vibration energy is transmitted back to the mounting equipment.

Mounted on a roadheader type undercarriage, this diesel driven impact ram can mine any shape opening in rock. In many urban and mining environments this machine has the potential to reduce, and in some cases, eliminate the need for explosives. It can complement the faster and more powerful tunnel boring machines by developing assembly chambers, stations, wide slots and other such openings in rock.

In principal, operation of the RAMEX Mining Boom is much like that of the hydraulic breaker. It is operator sensitive and when the ram is forced into rock the machine commences to fire and it runs until the pre-load force is removed or the rock is broken out.

CUTTING SLOTS IN ROCK

The Beus and Phillips report discussed "a two part rock breakage process: 1) the formation of a network of cracks in the rock, thus creating new free surfaces, and 2) the removal of chunks of rocks by forming secondary cracks reaching these free surfaces." (Figure 4) Their testing used the singlemoil or chisel point cutting tool. A free surface must exist or be created to effectively break solid rock. Sharp corners in the rock structure must be avoided to reduce stress failures in the rock cutting tool.

Using the same cutting action as do rolling button cutters for raise boring operations, the RAMEX kerfing rock cutter makes wide slots into the tunnel face. A series of 4 or 5 inch deep slots about 16 inches wide are cut with the kerfing cutter. (Figure 3) The cuttings produced are a little larger in size than those found in raise boring. Rock cores left between the slots are broken out using the mining boom swing force. This core breaking action produces very large rock cuttings. In tunneling gravity removes these cuttings from the face. This core breaking operation is a highly energy efficient method of removing rock.

To date RAMEX development work has concentrated on tunneling. The following discussion is based on early concept studies for shaft sinking and may or may not represent the final equipment operation.

Gravity is not expected to be of much help in removing mechanically cut rock from the bottom of a shaft. No attempt will be made to create or break rock cores between the wide kerf cut slots. High pressure water and/or air will be used to flush the cuttings from between the rock cutter edges. (Figure 4) Spoil will be flushed up immediately following the cutter. (Figure 5) The mining boom will hang in the shaft and pivot about the vertical center line. This action will leave a hemispherical shaft bottom that will be sunk in lifts of about eight inches. A lift will consist of spiraling the kerf cut slots in from the outside shaft gage to the center and back out again.

STATUS OF DEVELOPMENT

In January of this year RAMEX Systems rolled out the prototype and started testing in a hard rock quarry near the city of Monroe Washington. The quarry rock is a hard brittle and very abrasive felsite with an unconfined compressive strength of 35,000 to 40,000 psi. Mining the blast fractured rock face at a rate of under two horse power hours per ton the 100 horsepower has demonstrated its ability to mine a very hard rock face at rates up to 50 tons per hour. At full power in an undisturbed rock face it is expected to mine at a production rate in excess of 25 tons per hour.

A continuing research and development program using the most recent techniques in instrumentation and data acquisition is being carried out. This program will continue until a fully reliable and field proven machine can be commercially produced.

TABLE 1

OPERATION	ROCK		$\frac{HP-HR}{TON (avg.)}$
	TYPE	STRENGTH (ucs)	
Impact Shaft Sinking (Ref 2)	quartz monzanite	28,000 psi	1.3*
RAMEX Test Site (Ref 3)	felsite	30,000 psi	1.2*
Channel Tn'l TBM (Ref 4)	chalk	2,000 psi	4.3†
Faeroe Islands TBM (Ref 5)	igneous extrusive	18,500 psi	4.9†
Calaveras Power Tn'l TBM (Ref 6)	quartz schist granodiorite	9,800 to 29,800 psi	4.5† 6.7†

* Including 50% of working time for boom positioning.

† Based on reported average advance rate.

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FIGURE 1
A modern hydraulic breaker

Schematic RAMEX
Mining Boom

FIGURE 2

FIGURE 3
IMPACT KERFING CUTTER
Model IKC 101

FIGURE 4
PROFILE OF SHAFT BOTTOM MADE
WITH A KERFING CUTTER

FIGURE 5

Cutting action and fluid flushing of a shaft boring rock cutter mounted on the RAMEX Mining Boom.

DEVELOPMENT OF A HARDROCK EXCAVATING MACHINE

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DEVELOPMENT OF A HARD ROCK EXCAVATING MACHINE

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ABSTRACT

This paper describes the development and testing of the RAMEX excavation system. Incorporating a synergy of low velocity, high energy impact and kerf cutting, this machine offers a competitive alternative to drilling and blasting in hard rock. It is a boom mining machine that incorporates the features of a lightweight roadheader and it can mine nearly any shape opening in hard rock.

The RAMEX Tunneler has been developed to satisfy industry demands for a non-explosive, flexible, continuous and rapid underground rock excavation system. It will produce minimum ground vibration in the urban environment and allow the flexibility necessary for thin seam mining conditions. It will also reduce initial ground support costs and has a minimum impact on rock integrity.

Developed by RAMEX Systems with cooperative research provided by the U.S. Bureau of Mines, this system, could produce the next step in the continuing evolution of mechanical mining systems.

INTRODUCTION

Technology has evolved, in the last twenty years, to the point that hydraulic impact breakers are now common place for demolition and secondary breaking throughout the world.

In the early 1970's, Ingersoll—Rand and Joy each developed a prototype hydraulic breaker for mining and tunneling operations (Hamilton 1975 & Wayment, 1976). At the same time the Cementation Co., of Tucson AZ, in cooperation with the US Bureau of Mines, developed a method of shaft sinking that incorporated impact breakers (Bues, 1981). These early operations indicated both promise and problems. Once the initial problem of equipment reliability due to vibration was more or less

resolved, the testing indicated that mining a full face in rock with themoil point tool was not practical. Themoil point could break rock to pre-existing fracture surfaces, but would tend to shape the working face into a cone. The mining boom could not be positioned in such a way as to enlarge that cone. Aside from the problem of equipment failures caused by high frequency high amplitude vibration, the rock face cone problem remained pervasive. A tool was needed that could cut wide slots for both the tunnel gage and the working face. In addition, a better solution to the equipment vibration problem was needed.

Heavy duty hydraulic breakers were used in Italy in1988 to construct several major tunnels in soft sedimentary rock. The contractor Cogefar, advanced one project, the Capo Nero twin track high speed rail tunnel, at a rate of 10 m (33 ft) each 24 hour working day (Smith, 1988). Breakers mounting amoil point tool worked well in the soft and somewhat fractured rock. The Italian contractors clearly demonstrated safety, economy and efficiency of operation using impact mining on a soft rock tunnel face. Offering the flexibility of the more familiar drill jumbo, the relatively low initial cost of the mounted breaker allowed the use of two or more units on each project. The problems of mining hard rock, though still present, are dampened in the softer sedimentary formations.

Figure 1 Prototype mining ram mounted on a hydraulic excavator.

THE RAMEX TUNNELER

With the objective of mining hard rock, Mr. W.D. (Bill) Coski, a Seattle professional engineer with extensive experience in designing blast hole drills, rock cutters, tunnel and raise boring machines, designed and patented a low velocity impact ram mining system that would substantially reduce equipment vibration and enhance rock cutting efficiency (Coski 1982). He also designed and has a patent pending for a kerf cutting rock cutter that can create wide slots in both the tunnel gage and face (Coski 1989). This rock cutter will also produce the large size chips we are used to seeing in raise and tunnel boring operations.

As of December 1988 a full size working prototype was assembled (Fig 1 & 2). Mounted on a hydraulic excavator, this system is undergoing extensive performance and reliability testing. The purpose of this paper is to describe current research and test developments of the RAMEX Tunneler and to provide performance and cost estimating methods developed for its application.

Figure 2. Impact mining ram - initial tests on concrete.

PRINCIPALS OF OPERATION

Reducing impact velocity is the key to reducing high frequency equipment vibrations and high stress levels. Nearly all breakers available today have a piston relative to tool (metal-to-metal) impact velocity in excess of 6.1 m/s (20 ft/s). They use a high velocity lightweight piston to strike and drive the breaking tool into a rock surface (Fig 3). It is the metal striking metal operation in this tool that produces the high pitch ringing sound associated with the hydraulic breaker. (Black 1968 & 1969 & Fu 1968)

Figure 3. Conventional hydraulic breaker. Showing the piston to breaking tool metal-to-metal contact.

The RAMEX Mining Boom consists of a two-stroke cycle single cylinder free-floating piston diesel driven into an air cushion bounce chamber (Fig 4). The air cushion transfers piston energy to the ram and returns the piston for the next cycle. There is no metal-to-metal impact. On the 74.6 kW (100 hp) prototype mining boom this piston has a weight of about 317.5 kg (700 lbm). The inertia of the reciprocating ram provides the energy required to drive the cutter head into the rock at the end of each stroke. The ram impacts rock at a velocity of 3 m/s (10 ft/s), or less than half that of hydraulic breaker piston impact velocity. This cycle produces a blow energy of 7.450 J ($5,500 \text{ ft-lbf}$). The piston cycles at a rate of 700 to 800 blows per minute, which is slow idle speed for a diesel engine. Unlike the hydraulic breaker there are no hydraulic losses and the bulk of the combustion energy is transferred into rock breaking. The only loss is heat through the combustion chamber and sliding mechanical friction of the piston on the cylinder wall.

A steady thrust force is applied to the reciprocating ram through a hydraulically actuated spring mechanism. Thrust force and ram position are controlled manually by the operator. A hydraulically operated mechanism which senses thrust pressure is used to regulate engine throttle setting automatically according to the power demand. This system also prevents operation of the engine when there is no power demand. Without this feature, destructive internal forces could occur.

The rock cutter used on the 74.6 kW (100 hp) mining boom consists of a high strength alloy steel plate bolted directly to the ram. It has no bearings or seals (See Fig 5). It mounts a number of equally spaced cutting edges. For hard rock these edges

Figure 4. Schematic RAMEX Mining Boom. The piston hurls the ram mass into the rock with each cycle.

are made up of several vertical rows of tungsten carbide buttons, similar in shape and spacing to those of a rolling rock cutter used on a raise boring cutter head. For softer rock, these edges are similar in shape and spacing to those of a tunnel boring machine disc cutter. In operation the cutter attacks the rock face at a slight angle. The mining boom is positioned at the top of the tunnel face and sweeps down producing a slot up to 0,41m (16 in) wide and 0,1m (4 in) deep. This process is used to cut the tunnel gage. It is then repeated leaving wide rock cores spaced across the tunnel face (Fig 6). The cuttings produced in rock are similar in size and shape to those produced by raise boring rock cutters except that the wide rock cores fracture in to much larger chips. The spoil produced is 10% to 15% minus 0,25 m (10 in) and plus 0,1 m (4 in) with the balance grading down to dust. Bond's theory of comminution indicates that production of larger rock chips requires less energy (Bond, 1961). The RAMEX Tunneler is expected to be, horsepower for horsepower, somewhat more efficient at rock breaking than a conventional tunnel or raise boring machine.

Figure 5. Kerfing Cutter for Rock

Figure 6. Tunnel face slotting operation.

Any partial face mining method can generate severe dust problems. The impact mining boom is expected to generate dust at the same rate as a roadheader. Provision for a moderate pressure 20.7 mPa (3,000 psig) water jet assisted cutting system, developed at the U.S. Bureau of Mines, is being incorporated into both the mining boom and the rock cutter (Handewith, 1985 & Kovseck, 1988). Bureau of Mines testing indicated that the generation of airborne respirable dust could be reduced over 80% using a properly designed 20.7 mPa (3,000 psi) water-jet-assisted cutting system. The water-jet has the added advantages of flushing chips from the interface between the cutter and the rock and also, cooling the cutter. This will provide longer cutting tool life and more efficient delivery of energy to the rock.

The mining boom can be mounted on a conventional backhoe undercarriage for some demolition, secondary breaking and tunneling operations. However, many underground mining and civil construction projects require a specific machine mounting base. Such an undercarriage must be capable of the gathering, loading and conveying spoil at a rate in excess of any expected excavation rate. It must also maneuver in restricted locations and be capable of being lowered through small shafts and reassembled underground. Finally, the undercarriage must be capable of interfacing with existing and planned underground mining systems.

TEST PROGRAM

Testing does not always agree with theory. Determining the degree of, and reason for, departure from theory is test engineering. The RAMEX test program was undertaken in joint cooperation with the U S Bureau of Mines, Pittsburgh Research Center (PRC). The PRC is providing a continuing program that includes an engineering test team, sensors, and data acquisition equipment to monitor shop and field testing of the 74,6 kW (100 hp) RAMEX prototype tunneler. The BOM is also providing data reduction and analysis at the PRC computer center.

Identifying Sampling Plan

Purpose of Testing - The RAMEX tunneler uses a free floating piston to deliver impact energy to a ram mounted rock cutter. The piston is diesel driven. Conceptually, the design has been shown to be viable. Before a production machine is built, system parameters must be selected for the most efficient and economical operation.

The prototype machine is instrumented for various operating parameters that can be tested while cutting concrete or rock (Fig 7). It is important during this testing to accurately monitor these parameters and to compare them to machine cutting efficiency.

Cylinder Pressure and Piston Location Relative to the Cylinder - As in any diesel engine, the RAMEX piston must compress the mixture of intake air and fuel until the temperature is high enough to cause ignition. The pressure required before ignition can take place in a naturally aspirated diesel engine is 2.76 to 4.83 mPa (400 to 700 psi). To determine the pressures generated a 0 to 70 mPa (10,000 psi) pressure transducer is located at the combustion end of the cylinder. Pressure for each stroke of the piston is recorded.

To pressurize the combustion chamber and create the first diesel cycle, piston movement is initiated using a surge of compressed air. A governed volume of start-up air up to 2.76 mPa (400 psi) is released suddenly through a check valve. The in-rushing air forces the piston into the combustion chamber with sufficient energy to create diesel ignition.

Start-up air pressure is monitored with a 0 to 35 mPa (5,000 psi) transducer located in the high-pressure air line near the bottom of the cylinder. Start air pressure is varied to determine the best pressure for efficient ignition and is correlated with the combustion pressure.

Each time the piston moves away from the combustion end of the cylinder, fresh air is blown in through intake ports. This 34,5 to 70 kPa (5 to 10 psi) scavenge air system removes the products of combustion and replaces them with the fresh air needed for the next ignition cycle. A variation in scavenge air pressure can indicate an obstruction in the ports, and a 0 to 1,38 mPa (200 psi) transducer is placed in the system just before the air enters the cylinder to monitor this condition.

Figure 7. Bureau of Mines data acquisition system for the RAMEX Tunneler.

To quantify both the duration and pressure of each fuel injection cycle the injection push rod was instrumented with a strain gage bridge system.

Piston Location The piston is not attached to a crank shaft; its travel is constrained by combustion pressure at one end and by bounce chamber pressure at the other end of the cylinder. In the bounce chamber a cushion of air stops the piston before it strikes the end of the chamber and produces the return force for the next diesel cycle. There is no metal to metal impact.

To determine relative location of the piston at any time during the operating cycle, a 0 to 500 ohm potentiometer is mounted on a pinion shaft and driven by a rack mounted

to the piston. For calibration, the piston is manually moved to the extreme length of travel on both ends of the cylinder. It is then possible to determine the exact location of the piston at any time during the operating cycle. Pressure is then correlated with piston location to determine the bounce chamber compression, and compression ratios.

Machine Cutting Efficiency Cutting efficiency is determined by calculating the amount of energy used by a machine to extract a given volume of rock. Initial testing was done by breaking reinforced concrete with the RAMEX Mining Boom.

Blow energy to the rock is measured with a 200g accelerometer directly attached to the rock cutter. During future tests the machine will be used in a mining situation and the force data will be correlated with the volume of rock broken and/or removed. Studies will be conducted on both energy/ volume and energy/surface area of the broken rock. These data will be used to determine the operating efficiency of the ram and the breaking efficiency of the rock cutter.

Blow energy is reacted by the inertia force of the ram. This force is then reacted by the hydraulic thrust control system and transmitted back to the tunneler undercarriage. To determine the magnitude of vibration transferred to the body of the undercarriage, a triaxial accelerometer was mounted on the right side of the machine chassis just below and in front of the operators's cab. To date, all tests have been conducted with the ram in the vertical down position. Future testing will be conducted with the cutting head in both the horizontal and vertical positions to study the effects of gravity on force/energy transfer.

Data Collection and Analysis Early data analysis indicated that the principles of design and operation are valid. Figure 7 is a schematic illustration of the data collection system used by the Bureau of Mines and Figure 8 is a typical stripchart playback of recorded data. The accelerometer data indicates that energy absorbed by the undercarriage is insignificant. Figure 8 shows typical operational data for piston stroke, combustion pressure and fuel injection force.

PERFORMANCE AND ESTIMATING

To evaluate operation of the 74,6 kW (100 hp) prototype RAMEX Tunneler a small volume of concrete was broken during shop tests. Surface quarry testing, for secondary breaking and performance evaluation studies are commencing as this paper is being written and will be reported elsewhere.

A computer program was developed for estimating costs and performance of the RAMEX Tunneler equipped with a 300 kW (400 hp) mining boom and comparing these costs with, drill/blast, a tunnel boring machine, and a roadheader. The program assumes one heading machine used for all four excavation methods and all equipment is purchased new for this project. This is an estimated and idealized tunnel project with a heading section of 29,17 m² (314 ft²) consisting of four portal drives of 1,52 km (5,000 ft) each. Moving time from portal to portal is considered. The costs of muck handling, ventilation and primary ground support are assumed to be nearly the same

CODAS HARD COPY UTILITY

Original DATA FILE: C:\SCOPE\CALIB01.RTS

1/02/1989 11:50 AM

Print DATA FILE: calib01.hcu

1/03/1989 09:33 AM

COMMENT:

Channel #1	+/-Full scale:	7.88e+00 IN	Base line:	6.51e+00 IN
Channel #2	+/-Full scale:	8.54e-03 Volt	Base line:	3.71e-03 Volt
Channel #3	+/-Full scale:	8.54e-03 Volt	Base line:	3.91e-03 Volt

COMPRESSION: 8

Figure 8. Test data for;
a. Piston Travel
b. Combustion Pressure
c. Fuel Injection Force

for all four mining methods and are not considered in the following estimates. However, it must be noted that there will be a considerable cost saving on tunnel support and lining when using any three of the mechanical mining methods in this study.

The 300 kW (400 hp) RAMEX Mining Boom Instantaneous mining rate was calculated, using (3.b.) and (3.c.) below. The calculated mining rate is 0,68 m³ (0.89 yd³) per minute. This yields an instantaneous penetration of 1,46 m (4.8 ft) per hour. Allowing time for boom and machine positioning, support system delays, cutter changing and maintenance the mining system utilization is estimated to be 40% of the available shift time. The average advance in an eight hour shift is then 4,58 m (14.7 ft).

The tunnel boring machine performance was estimated using the procedures previously published in the Proceedings of the CIM and the Underground Mining Methods Handbook (Handewith, 1972 & 1980). The roadheader performance was estimated using the procedures previously published in the 1983 Proceedings of the UTRC. (Handewith 1982)

The computer program was designed with the following assumptions:

1. In conformance with U.S. standard practice all units are in U.S. english notation.
2. All capital costs are written-off over the length of the defined project. Equipment salvage is 20% to 25%. Insurance and interest are each estimated to be 1% of the capital equipment value each month for the life of the project. The program allows for an operating profit. Equipment delivery times are not considered in the completion schedule.
3. The rock strength is 20,000 psi (138 mPa) ucs, and moderately abrasive shale or dolostone with an in situ weight of 170 lb ft³ (2,723 kg/m³). This rock is considered ideal for a tunnel boring machine or drill/blast, but, would not normally be considered economical for a roadheader. It will require minimum temporary support and has:
 - a. Swell factor of 1.4 for drill/blast and 1.7 for the mechanical mining methods.
 - b. Crushing rate equation (Handewith, 1972)) of:

$$F = 5,000 \text{ lbf} + [(95,250 \text{ lbf/in}) \times \partial]$$

Where: F is the machine thrust force applied to the cutting tool, and ∂ is penetration of the cutting tool into the rock, in inches, each time force F is applied.

- c. Energy rate of mining is 4 to 5 hp/hr per ton.
4. Ground conditions - excellent, no major faults or water in-flows. Ground support costs are the same for all four excavation methods, even though drill/blast would generally be expected to require somewhat more support. The cost of and time to place primary ground support or final lining is not considered.
5. Tunnel cross-section of 314 ft² (29.7 m²). For a tunnel boring machine this is a 20 ft diameter circular section. It is a classic horseshoe section for the other three mining methods.

Table 1 lists the computer program input requirements and Table 2 lists the output data. Figures 9a to 9g provide a graphic display of the output data for the assumed tunnel project estimates. Using the RAMEX 300 kW (400 hp) tunneler the total cost of excavation is estimated to be 3% higher than drill/blast. Time to complete is estimated 12% faster than drill/blast. As stated above, the assumed rock conditions are much too hard for a standard roadheader application. The slow progress rate and high wear part cost preclude the competitive use of this machine. Once it is on-site, the tunnel boring machine is easily the fastest method of excavation. However, the high capital cost, inflexibility of tunnel shape and slow turn radius preclude its competitive use on many tunneling and mining projects. Allowing for modest errors in estimating assumptions the RAMEX Tunneler should be competitive with drill/blast methods.

Figure 9. Estimating program printout, 1988 U.S. dollars.

a. Total Cost of Excavation

b. Time to Complete Excavation

c. Capital Costs

d. Energy Costs

e. Expendables Costs

f. Labor Costs

g. Site Preparation Costs

CONCLUSION

The RAMEX Tunneler has been developed to satisfy industry demands for a non-explosive, flexible, continuous and rapid underground rock excavation system. It will produce minimum ground vibration in the urban environment and allow the flexibility necessary for thin seam mining conditions. It will also reduce initial ground support costs and has a minimum impact on rock integrity.

Incorporating a synergy of low velocity, high energy impact and kerf cutting, this machine offers a competitive alternative to drilling and blasting in hard rock. It is a boom mining machine that incorporates the features of a lightweight roadheader and it can mine nearly any shape opening in hard rock.

The RAMEX Impact Mining System is expected to be competitive, in both cost and performance, with that of conventional drill/blast excavation. As of December 1988 the 74,6 kW (100 hp) prototype RAMEX Mining Boom is undergoing shop testing for performance and reliability. It is expected to move to a surface quarry for secondary breaking tests in early 1989. Following the quarry tests we expect to be starting underground testing.

TABLE 1

PERFORMANCE ESTIMATING
PROGRAM INPUT PARAMETERS

- a. Capital Costs
Equipment Cost
Initial Spares & Cutters
Backup Gear
- b. Productivity
Average Length of Drift - feet
Number of Drifts
Penetration Rate - ft/hr (instantaneous)
System Utilization - %
Assembly and Checkout Time - shifts (initial)
Time per Move - shifts
Take Down Time - shifts
- c. Energy Costs
Installed Power - kW
Electricity @ __ ¢/kWh
Diesel Fuel Consumption - gal/hr
Diesel Fuel @ __ ¢/gal
- d. Labor Costs (estimated at \$/hr)
Machine Operator(s) or Drillers
Mechanic/Operator(s)
Driller helpers
Walking Boss
Laborers
- e. Site Preparation Costs (for excavating machine placement only)
Initial
Each Move
- f. Expendable Item Costs
Cutting Tools - \$/yd³
Spare Parts - \$/yd³
Conveyor Parts - \$/yd³
Explosives Consumption - \$/yd³
Explosives Cost - \$/lb
Drilling Expendable Items - \$/yd³

Table 1. Performance estimating program input data.

TABLE 2

**PERFORMANCE ESTIMATING
PROGRAM OUTPUT PARAMETERS**

Program Outputs are:

- a. Capital Costs
 - Total Interest @__% per month (for the life of the project)
 - Total Capital & Interest
- b. Productivity
 - Total Drive - feet
 - Average Advance - ft/shift-hour
 - Boring Hours to Complete
 - 8 hour Shifts to Complete
 - Number of Portal Moves (including removal from last drift)
 - Total Moving Time - Shifts
 - Revised Productivity, Avg feet/shift
 - Working Shifts to Complete
 - 3 Shift Days to Complete
 - 21 Day Months to Complete
 - Total Years to Complete
- c. Energy Costs
 - Hours @ 100% power
 - Hours @ 30% power
 - Total Kilowatt Hours (kWh)
 - Diesel Fuel @ ____¢ per gallon
 - Total Energy Cost
- d. Labor Costs
 - Total Workers per Shift
 - Total Worker Shifts
 - Total Labor Cost
- e. Expendables - Cost
 - Total Excavation - Cubic Yards
 - Total Expendables - Cost
- f. Sub Total Project - Cost
 - Total US Dollars
 - Dollars per Foot of Drift
 - Dollars per Cubic Yard
 - Dollars per Ton

Table 2. Performance estimating program output data.

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DEVELOPMENT OF AN IMPACT CUTTING
TOOL FOR ROCK

by

Henry Harling Haselton

A thesis submitted in partial fulfillment
of the requirements for the degree of

Master of Science in Civil Engineering

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to Offer Degree

Department of Civil Engineering

Date

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Abstract

Development of an Impact Cutting
Tool for Rock

by Henry Harling Haselton

Chairperson of Supervisory Committee: Professor Teresa A. Taylor
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The objective of this research was to determine if any differences exist between low impact velocity/high cutter mass and high impact velocity/low cutter mass impacts on the effectiveness of the RAMEX hard rock cutting tool. The impact velocity/cutter mass combinations were adjusted to deliver a constant impact energy. This was done to evaluate whether the RAMEX cutter would perform comparably on two different impact delivery systems which utilize different impact velocities and cutter masses for impact. These systems include the RAMEX mining boom and the conventional hydraulic hammer.

To accomplish this objective, a series of laboratory tests were performed in which a test cutter similar to the RAMEX cutter was dropped on specimens of concrete and rock using a drop test fixture located at the U.S. Bureau of Mines Pittsburgh Research Center. The test specimens consisted of low and high strength mortar (approximately 3000 and 10,000 psi, respectively), Barre Granite and Berea Sandstone.

The measurements used to evaluate the effectiveness of the cutter under the different impact velocity/cutter mass combinations included instantaneous cutter penetration, specific energy, cutter force, cutter vibration, impulse, and coefficient of restitution. Based on the test results, it was concluded that instantaneous cutter penetration was the best laboratory measurement for identifying the effectiveness of the cutter. The effect of impact row spacing on cutter efficiency was also tested.

The results of the tests showed no significant differences in the depth of instantaneous cutter penetration under the different impact velocity/cutter mass combinations. A number of general trends were identified from the other measurements that could be pursued in future research efforts.

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CHAPTER 1 INTRODUCTION

The development of technology to mechanically excavate *hard* rock has been successful only to a limited extent. Both the civil construction and mining industries have long recognized the need for a method to mechanically excavate hard rock, which has led to the development of the tunnel boring machine (TBM) and other mechanical excavation methods. The TBM, while capable of boring relatively straight, circular tunnels, is not highly mobile, and is incapable of excavating non-circular underground openings. Other, more mobile machines such as the roadheader, continuous miner and hydraulic breaker are capable of excavating openings of almost any geometry, but have not been effective in hard rock. Until recently, most hard rock excavation has been the domain of explosives, which is an excavation method with limited potential for mechanization. Explosives also have undesirable side effects, such as noise and vibration, that limit their use in urban environments.

To fill the demand for a mobile, mechanized method of excavating hard rock, the RAMEX impact mining boom and cutter system was developed in the late 1980s. As presently designed, this system is capable of cutting up to 8 in. wide vertical slots, or kerfs, in a rock face using rapid (up to eleven times per second) impacts and a steel cutter equipped with parallel rows of tungsten carbide button indentors (Figure 1-1).

Between the kerfs, a ridge of rock is left that can be easily removed with cutter impacts (Figure 1-2). During operation, the resulting rock fragments vary from dust size to approximately one foot in length. Typically, rock fragments produced beneath the cutter are thin "chips" with a maximum size of about 3 in., while the larger fragments are produced by impacting the unconfined ridge. Thus, the RAMEX excavation method does not remove rock by pulverization, but by controlled fracturing of the rock using impact.

The RAMEX mining boom delivers impact energy to rock by means of low velocity/high mass impacts. The prototype of the RAMEX system was successfully field tested in 1990, using the prototype impact cutter design shown in Figure 1-1. With the effectiveness of the concept of the RAMEX mining boom impact delivery system proven, RAMEX is currently directing its research and development to the design of the cutter.

The prototype RAMEX mining boom utilizes moderate impact energy of about 5,500 ft-lb per impact. This energy is delivered utilizing a 3500 lb mass and a velocity of about 10 ft/sec. Other impact systems

currently exist and are widely used in the industry, such as the hydraulic breaker (Figure 1-3). Typical hydraulic breakers operate at approximately the same frequency as the RAMEX mining boom, but utilize low mass/high velocity to deliver the impact energy. A typical 5,500 ft-lb hydraulic breaker, for example, delivers impact energy at a velocity of about 20 ft/sec. with an impact mass of about 900 lb. Although the range of impact energies utilized by hydraulic breakers varies (approximately 150 - 50,000 ft-lb), most breakers deliver impact energies comparable to the RAMEX mining boom.

FIGURE 1-1 Prototype RAMEX Kerfing Cutter
(Source: Handewith et al., 1989)

As part of its research and development, RAMEX has funded this research to study the effectiveness of its cutter design when used with a high velocity/low mass system such as the hydraulic breaker. At present, hydraulic breakers utilize "moil point" or chisel-type cutters (Figure 1-3), which are effective in breaking weak rock, concrete, and unconfined hard rock such as loose boulders. When attacking a very hard

confined rock, however, themoil point or chisel cutter creates a cone-shaped opening in the rock mass, which eventually cannot be advanced.

FIGURE 1-2 Rock Cores Left Between Slots on Excavation Face
(Source: Handewith et al., 1989)

The primary objective of this research was to evaluate the effect of low velocity/high mass (RAMEX) vs. high velocity/low mass (hydraulic breaker) impact energy on the ability of the RAMEX cutter to break confined concrete and rock specimens. This was done to evaluate the feasibility of combining the existing hydraulic breaker delivery system technology with the RAMEX cutter technology for excavating in hard rock and concrete.

The tests were accomplished using a *drop tower testing device* located at the U.S. Bureau of Mines Pittsburgh Research Center (PRC) (Figure 1-4). In the tests, a simplified version of the RAMEX cutter was attached to a vertical slide, and dropped from a pre-determined height onto specimens of concrete and rock of different strengths. The impact energy used in the drop tests was about 250 ft-lb per button indenter, which approximates the impact energy of the prototype RAMEX mining boom. The test cutter

contained a single row of three tungsten carbide button indentors, spaced at 3 in. centers (Figure 1-5). Two impact velocities were used (approximately 10 and 20 ft/sec), and the masses used were 485 and 130 pounds, respectively, to achieve the desired total impact energy of 750 ft-lb. One row of button indentors was utilized so that impact row spacing could be varied to evaluate an optimum row spacing for the cutter design. The method used to evaluate the difference between the two impact modes included measurements of force and penetration during drop tests. The energy required to remove a unit volume of rock during impact (specific energy) was also measured.

Moll	
Blunt	
Chisel	

(a) Schematic - Typical Hydraulic Breaker

(b) Common Tool Shapes

FIGURE 1-3 - Hydraulic Breaker
(Source: Anderson et al., 1989)

The following tasks were completed in the testing program:

- Investigation and review of readily available, current literature (since 1990) concerning rock cutter technology, and research related to rock fracture.
- Development of concrete test specimens for use in drop tests.

- Laboratory testing of the cutter using a drop tower testing apparatus, with preliminary tests being performed at the University of Washington, and final tests being performed at the Pittsburgh Research Center (PRC).
- Evaluation of laboratory test results

A summary of mechanical excavation methods for hard rock is presented in Chapter 2 along with some discussion of the factors that affect rock cutting performance. Chapter 3 presents a general review of current literature believed pertinent to this study. Details of the investigative methods for the laboratory testing program are presented in Chapter 4, and Chapter 5 discusses the results of the investigations. Chapter 6 presents a summary of the test results, conclusions, and recommendations for related research in the future. Appendix A presents the development and strength testing of the concrete test specimens. Appendices B and C present data summaries that were too lengthy to incorporate into the main body of text.

FIGURE 1-4 - Drop Testing Device

FIGURE 1-5 - Impact Cutter for Laboratory Tests

CHAPTER 2

MECHANICAL EXCAVATION METHODS FOR HARD ROCK

A large body of literature is available on mechanical excavation methods, a sampling of which was reviewed for this study. This chapter discusses some of the factors that affect the performance of hard rock mechanical excavation machines, the evaluation of the performance of these machines, and descriptions of some of the specific mechanical methods used for excavating hard rock. This literature review was undertaken to better understand the RAMEX impact cutting concept, and its relationship to other more common rock excavating technologies. The information in this chapter is also intended to provide the civil/geotechnical engineering audience with some background on the underground excavation industry, and the potential applications for impact cutting technology.

The majority of literature on hard rock excavation falls into two general categories: 1) analytical studies (including computer modeling) rock excavation using roller disc and drag pick methods [e.g. Whittaker et al., 1992; Fowell, 1993]; and, 2) performance prediction/evaluation of tunneling machines as case studies [e.g. Nelson et al., 1991; Nelson et al., 1993; Nilsen and Ozdemir, 1993; Rostami and Ozdemir, 1993]. Very little current literature was located that investigates impact methods for rock excavation.

2.1 PERFORMANCE FACTORS FOR MECHANICAL EXCAVATORS

The selection of a mechanical hard rock excavation tool and the prediction of its performance depend on several factors, the most important of which are the physical properties of the rock to be excavated. These include the rock strength, and fracture characteristics. There is a wealth of literature describing these properties and the testing methods used to evaluate them [e.g., Atkinson, 1987; Farmer, 1968; Hoek and Brown, 1980; Obert and Duvall, 1967; Wahlstrom, 1973; Walton, 1958; Williams, 1952; Whittaker et al., 1992; Atkinson, 1993; Brook, 1993; Pells, 1993; Szymanski et al., 1989; Inyang and Pitt, 1993; Nelson, 1993].

There is no excavation machine that works efficiently in all rock materials. Since appropriate excavation machines should first have the physical ability to efficiently fragment the rock, the ability of the rock to resist fragmentation should be defined before a machine is selected. Laboratory and field tests are commonly performed to classify rock, and to evaluate a machine's potential effectiveness. Some methods

for performing these evaluations are referenced in this section, and some definitions for describing rock properties are presented.

Rock is mechanically excavated by imparting energy to a rock mass through a cutter, resulting in failure of the rock mass. Many factors affect the ability of a cutter to fracture rock, but its in-situ strength is the main factor [Hoek and Brown, 1981; Bullock, 1994]. The most common index of rock strength is its unconfined compression strength (UCS). The use of UCS as a strength index for excavation, however, is controversial since rock is neither unconfined in-situ, nor does it fail entirely in compression when being subjected to mechanical excavation methods [Nelson et al., 1991; Nelson et al., 1993]. Furthermore, there can be "scaling effects" (i.e. the effect of the test specimen size) and rock mass factors, which cause the performance estimates from UCS values to differ from the actual performance of the machine.

The use of UCS as a strength measurement for mechanical excavation probably stems from other rock engineering experience, where UCS is used for the design of foundations on rock. In this case, the engineer is interested in the ability of a rock to *withstand compressive loads*, making the compressive strength (though not necessarily the UCS) appropriate for the failure mode of interest. In mechanical excavation technology, however, one is interested in *the ability of the rock to fracture in tension*, and tensile strength of rock does not necessarily correlate with UCS.

The adoption of UCS as a strength index for rock excavation is analogous to the use of Standard Penetration Test (SPT; ASTM D-1586), which has been adopted in soil mechanics practice for a large variety of applications, not all of which are necessarily appropriate. The fact remains that UCS, like SPT, is a relatively simple and economical test to run, and there is a large database of information that can be used by designers for excavation applications. Because of this, many performance prediction methods for rock excavation use UCS as a basis [Ross et al., 1972]. In addition, UCS shows good correlation with TBM performance in many conditions [Dollinger, personal communications, 1995]. While using UCS as a measure of rock borability has its drawbacks, it does provide a *relative* strength measurement for rocks, and has been used in this study to define rock strength, as shown in Table 2-1.

As mechanical excavation technology has become more sophisticated, so have the methods to predict performance of machines. Many investigators [e.g., Inyang and Pitt, 1993; Nelson, 1993; Nilsen and Ozdemir, 1993; Rostami and Ozdemir, 1993] have developed excavation performance prediction methods that range in complexity and accuracy. Some are based on tests specific to the proposed excavation technology (e.g. linear cutting tests for TBM performance, and rock fracture hammer tests for impact

cutting methods), and others use standard rock tests such as UCS and Brazilian tensile test in an attempt to approximate the failure mode of interest. Still, others use tests that measure rock strength indirectly along with empirically derived correlations to predict machines performance (e.g. the point load test and Schmidt hardness test).

Table 2-1: Deere & Miller's Classification of Intact Rock Strength

Description	Unconfined Compressive Strength		Examples of Rock Type
	psi	Mpa	
Very Strong (Very Hard)	>30,000	>200	Quartzite, gabbro, basalt
Strong (Hard)	15,000 - 30,000	100 - 200	Granite, gneiss
Moderate	7,500 - 15,000	50 - 100	Sandstone, slate
Weak	3,500 - 7,500	25 - 50	Coal, siltstone
Very Weak	150 - 3,500	1 - 25	Chalk, rocksalt

As cited by Hoek & Brown, 1980

A recent summary of standard rock strength testing methods is found in a paper by Lade [Lade, 1993]. This paper discusses the factors that influence the measurement of rock strength, and different failure criteria for rock. Another paper on the measurement of rock strength [Brooks, 1993] discusses both of direct and indirect strength measurements. Nelson [Nelson, 1993] provides a good review of rock properties and testing methods used in TBM performance prediction.

2.2 PERFORMANCE MEASUREMENT OF MECHANICAL EXCAVATION METHODS

A large body of literature on the assessment of mechanical excavation performance is available [e.g., Nelson, 1993; Nilsen and Ozdemir, 1993; Rostami and Ozdemir, 1993]. It is important to have a consistent method of evaluating the cost effectiveness, reliability, and efficiency of different excavation technologies. One method of evaluating the performance of mechanical excavation machines is through specific energy (SE), i.e. the energy required to remove a unit weight (or volume) of rock. The units of SE that are typically used in the underground excavation industry are hp-hr/ton or hp-hr/yd³.

$$SE = \frac{E}{W} \text{ or } \frac{E}{V} \quad (2-1)$$

where:

E = Energy used for Excavation (e.g. hp-hr)

W = Weight of Excavated Material (e.g. ton)

V = Volume of Excavated Material (e.g. yd³)

SE of an excavation machine is primarily a function of rock properties and machine characteristics (primarily cutter geometry). For a given geologic unit, the performance of different excavation machines may be compared based on SE. A lower SE value indicates a more efficient excavation system than a higher SE value for a given rock type because less energy is used in the removal of a unit weight (or volume) of rock. It is important to note that SE does not measure actual production or advance rates, since these are influenced by factors such as worker proficiency, machine down time, and adverse geologic conditions. In general, however, SE is inversely proportional to production rates.

It has been observed that low SE results when large rock fragments are generated during excavation [Mauer, 1968; Whittaker et al., 1992; Handewith et al., 1989; Handewith et al., 1990; Linesberry, 1991; Nelson et al., 1991; Nelson et al., 1993]. This is because small fragments result in a larger total surface area of the excavated rock, and therefore require more energy during breakage. Figure 2-1 illustrates this effect, and also illustrates that the relationship between specific energy and average fragment size is non-linear. It is worth noting that while larger fragments are desired for mechanical excavation methods to increase efficiency, the maximum size may be limited by the capability of machinery to remove such fragments.

FIGURE 2-1 - Specific Energy Relationship to Final Particle Size (after Mauer, 1968)

It is difficult to compare different excavation technologies using SE, especially when it is reported for different geologic and machine conditions. For this reason, it is best to indicate SE along with other of rock properties, such as UCS when comparisons are made. Usually, SE is reported as a range for a particular machine in order to account for variations in rock conditions.

Mechanical excavation methods have also been reported in terms of instantaneous penetration [Nelson, 1993]. Deep penetration of the cutting tool into the rock generally represents more efficient fragmentation of the rock mass and lower SE values. This observation is supported by research using static penetration tests [Detournay et al., 1995, Tan et al., 1994], and also by the evaluation of field data for a variety of mechanical excavation methods [i.e., Fowell, 1993; Sun et. al, 1992].

2.3 MECHANICAL EXCAVATION TECHNOLOGIES

Recent summaries of mechanical excavation methods for hard rock are available in the literature [e.g., Bullock, 1994; Campbell, 1989; Friant and Ozdemir, 1993; Murray et al., 1994; Nelson et al., 1991; Nelson et al., 1993; Pathak and Udd, 1993]. Most of the recent literature on mechanical excavation of hard rock is primarily concerned with tunnel boring machines equipped with disc cutters. Some recent literature also discusses impact cutting methods for hard rock, primarily in the context of hydraulic breakers [Anderson et. al., 1989; Armando et. al, 1989; Bauer, 1994; Smith, 1988; Waymont et al., 1976]. No literature was located concerning the rock breaking concept used with the RAMEX system, although some publications concerning field studies of the RAMEX system exist [Handewith et al., 1989; Handewith et al., 1990; Handewith et al., 1994; Kovscek et al., 1991; Linesberry, 1991].

As previously mentioned, the industries primarily concerned with excavation of hard rock are mining and civil/construction. While the reasons for creating underground openings in these industries may be different, the methods of fragmenting the rock to create the underground openings are similar. The geometry of the underground openings are also often different in mining than in civil/construction applications, so the configurations of the excavating machines usually differ. Prior to the development of mechanical rock breaking methods, drill and blast technology was the main method of hard rock excavation. This technology has some disadvantages, including safety problems, a low potential for automation, and it results in damage to the rock mass surrounding the excavation. Drill and blast

methods, however, have been improved, and are still frequently used for many underground excavation projects. No single mechanical excavation method available today can perform effectively under as wide a range of conditions as drill and blast methods have. Because of the disadvantages of drill and blast technology, however, there is growing demand for an economical, versatile and mobile mechanized excavation system for hard rock.

Mechanical excavation methods have many advantages over drill and blast methods. These include:

- increased safety
- increased automation
- continuous tunnel advancement and muck (excavation spoils) removal
- minimal rock over-break
- reduced disruption of surrounding ground
- reduced ground support requirements
- precise control of opening geometry

Detailed reviews exist in the literature for hard rock excavation tools [e.g., Linesberry, 1991; Bullock, 1994; Friant and Ozdemir, 1993; Pathak and Udd, 1989]. The most common continuous underground excavation machines for hard rock include full-face TBMs, raise boring machines, mobile miners (using disc-cutter technology), and hydraulic breakers. Table 2-2 lists current mechanical excavation methods for hard rock, showing the rock strength range of application, and some of the advantages and disadvantages of each method.

In the civil/construction industry, the majority of underground construction projects are tunneling related. These projects often require excavations of circular geometry and have long, straight reaches, for which TBMs are well suited. For TBMs to be cost effective, however, they must be used in long tunnels due to their high capital and mobilization costs.

The mining industry rarely uses circular openings other than for access, and often requires non-circular excavations to access or follow concentrations of minerals. Non-circular excavations are also required in some civil/construction applications and require machinery that is capable of creating such openings. Such openings are generally short in length, making high capital cost machines (such as TBMs) inappropriate.

TABLE 2-2 - General Overview of Mechanized Hard Rock Excavation Methods

METHOD	TYPICAL ROCK UCS APPLICABILITY	ADVANTAGES	DISADVANTAGES
Tunnel Boring Machine (TBM)	Up to 30,000+ psi; SE = 3-6 hp-hr/ton	Efficient, cost effective for large jobs, with constant opening geometry and relatively straight reaches	Not mobile, high initial capital costs, very long turning radius, creates constant opening geometry only
Revolving Wheel Disc Cutting Machines (Robbins Mobile Miner)	Up to 30,000+ psi	Relatively mobile, uses proven disc cutting technology	Expensive, maintenance problems, some limitations in excavation geometry
Undercutting using Disc Cutters	Up to 45,000 psi	Relatively mobile, energy efficient, utilizes rock's low tensile strength for cutting, produces large fragments (low SE)	Opening geometry limitations, new technology - not industry-tested
Minidiscs	Up to approximately 30,000 psi	Low thrust, small turning radius, energy efficient	Under development
Roadheaders / Continuous Miners	up to 15,000+ psi; SE = 15 - 20 hp-hr/ton	Mobile, economical, variable geometry possible, may be fitted with minidiscs for harder rock	Not effective on hard rock applications
Hydraulic Breakers with High Energy Hydraulic Hammers (HEHH)	30,000+ psi	Mobile, inexpensive, infinite opening geometries possible in soft or highly fractured rock; more commonly used for secondary rock breaking	Not effective for intact hard rock masses; face advances to a "cone" in hard rock

These requirements for economical excavation of non-circular openings sets the stage for a highly flexible, mobile, hard rock excavation tool such as the RAMEX system. While it is doubtful that the RAMEX excavation system can compete with TBMs in cost efficiency in long tunnels, it could compete with TBMs for smaller tunneling projects where high capital costs and machine mobility are a factor. The RAMEX system may also be used in conjunction with TBMs for the excavation of alcoves, cross-cuts, and other openings of varied dimensions.

2.4 BACKGROUND OF THE RAMEX IMPACT CUTTING SYSTEM

A comprehensive evaluation of the RAMEX excavation system and its development was prepared by Linesberry [Linesberry, 1991]. This evaluation includes a full description of the RAMEX system, a history of previous testing programs, and a comparison of the RAMEX system with other excavation technologies. Other reports on the development of the RAMEX system have been published [Handewith et al., 1989; Handewith et al., 1990; Kovscek et al., 1991].

The RAMEX impact delivery system consists of a two-stroke, single cylinder, free-floating piston, that is driven into an "air-cushion" chamber (Figure 2-2). The piston is diesel powered, and the prototype system delivered about 100 hp while cycling at a rate of 700 - 800 blows per min. Over 90% of the combustion energy is transferred to the rock, which is far greater than most conventional mechanical excavation methods. An air cushion transfers the impact energy to a ram, and returns the piston for the next cycle, without metal-to-metal contact. The impact cutter, with multiple rows of tungsten carbide bits (Figure 1-1), is attached to the ram. The 3500 lb ram and cutter impact the rock face at a velocity of about 10 ft/sec, resulting in an impact energy of about 5500 ft-lb.

The RAMEX mining boom can be mounted on any undercarriage of 18 ton or greater, such as conventional backhoes and roadheader carriages. With such an undercarriage, the RAMEX mining boom is mobile, and capable of excavating openings with tight turning radii and variable geometries.

FIGURE 2-2 - Schematic Prototype RAMEX Impact Mining Boom and Cutter
(Source: Handewith et al., 1989)

As explained in the introduction, the cutter produces slots, or kerfs the width of the cutter itself (about 8 in.) and up to 4 in. deep into the rock face. The slots are typically cut vertically from top to bottom of the rock face, and parallel to one another (Figure 1-2). High pressure water jet assists can be used to suppress dust, and to assist in the removal of rock fragments from beneath the cutter. This reduces energy loss due to re-crushing of rock fragments. An unconfined ridge of rock is left between the slots, which is later

easily removed by impacting with the cutter. The removal of the unconfined ridge produces very large (up to one foot in the largest dimension) fragments of rock.

The 1989, field tests of the prototype RAMEX system were conducted at a quarry near Monroe, Washington on basalt/andesite with a UCS of 30,000+ psi. Production rates at this site were approximately 6 yd³/hr. and the SE was approximately 5.75 hp-hr/ton [Kovscek et al., 1991].

With the development of the RAMEX mining boom well under way, research and development is currently being directed toward the cutting tool itself. It is believed that the RAMEX cutting tool would perform well if mounted on a conventional hydraulic breaker. If this can be demonstrated, it would allow marketing of the RAMEX cutter separately from the mining boom. The objective of this study was to test the feasibility of using the RAMEX cutting tool on a hydraulic breaker using laboratory impact studies.

2.5 BACKGROUND OF THE HYDRAULIC BREAKER SYSTEM

A hydraulic breaker can be mounted on almost any backhoe or excavator undercarriage. Rock breaking is accomplished by direct transfer of impact energy from the hydraulically driven hammer and piston to the rock. In contrast to the RAMEX system, the hydraulic breaker utilizes high velocity/low mass impact to derive its impact energy. Also, with the hydraulic breaker, the impact energy is delivered by metal-to-metal contact between the piston and the cutter. This results in high dynamic loads and high equipment wear. To fracture the rock, a "threshold energy" must be delivered, and the hydraulic breaker must be sized so that it delivers an appropriate amount of energy based on the rock conditions.

Common cutter tips used with hydraulic breakers include themoil, chisel, or blunt tips (Figure 1-4). Chisel tips are generally used for underground rock breaking, while the blunt tip is typically used for secondary rock breaking. Themoil point is used only on smaller hydraulic breakers with low impact energy.

A number of authors have documented case histories involving hydraulic breakers [e.g., Anderson et al., 1989; Armando et. al, 1989; Bauer, 1994; Smith, 1988; Waymont et al., 1976]. These reports are primarily concerned with performance evaluation and prediction. Published research into the cutting principles for these machines, however, appears to be few.

Hydraulic breakers became popular in the early 1970s, primarily for secondary rock breaking and concrete demolition. Early models of hydraulic breakers lacked sufficient impact energy to effectively break rock in underground applications. With the development of high energy hydraulic hammers (HEHH), however, full-face tunneling with hydraulic breakers has become possible [Anderson et al., 1989; Armando et. al, 1989; Bauer, 1994; Smith, 1988]. The use of HEHH in full-face tunneling, however, has been rare, with only a few successful applications being reported. Generally, the rock must be both relatively soft and highly fractured for successful use of hydraulic breakers equipped with HEHH in tunneling applications. This is because in non-fractured, hard rock, hydraulic breakers equipped with HEHH cut a cone at the tunnel face, and require alternate excavation methods to advance [Linesberry, 1991].

The RAMEX cutter, on the other hand, has been found not to produce a cone when excavating in hard, intact rock. Thus, if the RAMEX cutter can be proved to be effective on a HEHH impact delivery system, the HEHH may then become an effective tool for excavating openings hard, intact rock, without coning of the rock face. The combination of the RAMEX cutting tool with HEHH technology, if successful, could represent a major advancement in hard rock excavation that would be available to the many contractors and equipment suppliers who already own HEHHs.

CHAPTER 3

LITERATURE REVIEW

The literature review is divided into three general categories: 1) properties of concrete as a testing medium for impact testing; 2) rock and concrete fracture behavior under an indenter; and, 3) fracture behavior due to impact. For the most part, only literature published after 1990 has been included. Literature reviewed in the context of mechanical excavation methods has been summarized and discussed in Chapter 2.

3.1 CONCRETE AS A TESTING MEDIUM

The test specimens used in the present study were composed of relatively fine aggregate concrete (mortar) so that the cement matrix strength would dominate the impact-induced fracture process, thus better approximating the behavior of rock. Mihashi et al. [1992] report that aggregate grain size may significantly influence the fracture behavior of concrete. As aggregate size increases, fracture toughness also increases if the concrete matrix strength is less than that of the aggregate [Whittaker et al., 1992]. This is attributed to meandering of the cracks during failure, which results in increased crack surface area, and due to this, increased energy required to fracture the material. Using finer aggregate subsequently results in fracture behavior and other physical properties that approaches homogenous material, where such crack meandering is not frequently observed [Green, 1958].

Standard methods for designing concrete mixtures are well known and are explained in the Portland Cement Association Handbook [PCA, 1990]. This reference describes all of the ingredients of concrete and how each ingredient influences the overall properties. It also discusses standard testing methods for concrete as suggested by the American Society of Testing and Materials (ASTM). Appendix A of this thesis discusses the properties of the concrete used for this study and the testing methods used.

3.2 ROCK FRACTURE BEHAVIOR UNDER AN INDENTOR

A large body of literature exists for fracture mechanics of brittle materials under static and dynamic loading [e.g., Atkinson, 1989; Whittaker, 1992; Guo, 1994]. More detailed description of the mechanics of impact damage on an idealized brittle medium is also presented in many references [e.g., Goldsmith,

1960; Liaw, 1983]. Relatively few investigations of fracture mechanics in the context of mechanical excavation, particularly with regard to impact methods, have been done. Whittaker et al. [1992] provides one of the most recent and up-to-date references in this respect.

The subject of rock failure under static loaded indentors has been well studied. Many of these studies were reviewed for background information, and two recent works [Detournay et al., 1995; Tan et al., 1994] are summarized below. This topic is of particular interest because the RAMEX cutter utilizes hemispherical tungsten carbide indentors arranged in parallel rows (Figure 1-1). During impact, these indentors are pressed into the rock, resulting in the development of near surface fractures similar to that observed in static tests on indentors of various geometries. It has also been shown that the cracking phenomena associated with a wedge indenter is similar to that of a button indenter [Whittaker et al., 1992].

The first of these studies [Detournay et al., 1995] describes an analytical model of the events that take place during normal indentation of rock by a wedge, including the development of a crushed rock zone beneath the wedge and the initiation of tensile fractures at the crushed rock/intact rock interface (Figure 3-1). Detournay et al. demonstrate that the crushed rock beneath the indenter is a transition zone between the compressive stress field immediately beneath the indenter, and the triaxial stress field at the crushed rock /intact rock interface. The highest tensile stresses in the intact rock occur at that interface, directly beneath the tip of the indenter. The initiation of tensile failure at this interface occurs when the stresses in the crushed rock zone reach a critical value, resulting in tensile forces large enough to cause opening of pre-existing flaws within the material

These tensile cracks typically include primary (median) cracks, that propagate in the direction of the applied force and secondary cracks that range from sub-parallel to parallel to the free surface (Figure 3-1). The cracking pattern observed in static indenter tests is believed to be similar to that which occurs during dynamic loading by an indenter, especially if the impact velocity is significantly lower than the sonic velocity of the rock [Tan et al., 1994; Liaw, 1983]. Loading rates for percussive drilling and other impact excavation methods (including the RAMEX and hydraulic breaker systems) are typically well below the sonic velocity of rock. The secondary cracks are primarily responsible for the formation of "chips", or thin rock fragments that occur between adjacent indentations [Whittaker et al., 1992]. The formation of these chips was observed during the field testing of the prototype RAMEX system [Handewith et al., 1989; Handewith et al., 1990], and is illustrated in Figure 3-2.

FIGURE 3-1 - Normal Indentation of a Wedge into Rock
(Source: Whittaker et al., 1992)

FIGURE 3-2 - Rock Fracture Pattern and Chip Formation Due to Adjacent Indentations
(Source: Kovcssek et al., 1991)

The second study [Tan et al., 1994] involved the static application of a hemispherical indenter on granite and marble specimens. In that study, investigations were made of rock failure and fracture processes caused by the indenter. It was concluded that conventional strength theories (e.g., used for typical rock engineering design) are inadequate for describing indentation damage to rock. As a result, modeling the subsurface median crack propagation using a fracture mechanics approach was attempted, and the effects of factors such as applied indenter force, indenter shape, and rock strength on the formation of this crack were examined. A "threshold" indenter stress was found to be required to initiate fractures under the indenter. Stresses below this threshold result in essentially elastic deformations. The fractures caused by a hemispherical indenter consisted of primary (median) cracks, secondary cracks, radial cracks, and cone cracks. The secondary and radial cracks interacted to form flat chips. Both the physical observations and model prediction of rock indentation fractures by Tan et al. indicate that the subsurface median cracks initiate outside the crushed zone, and develop radially away from the indenter. Tan et al. suggest that the main rock fragment removal mechanisms in percussive drilling and boring methods is the formation of these chips.

3.3 FRACTURE DUE TO IMPACT

The concepts identified in the static indenter tests are useful in identifying the stresses in the rock that are imposed by an indenter. While the observation of rock failure under an indenter provides important insights for this study, the effect of impact loading is also important. The dynamics of impact loading on "semi-infinite" targets has been explored in many studies [e.g. Goldsmith, 1960; Fuchs, 1963, Liaw, 1983]. Just as in the static indenter tests, it has been found that there is a threshold impact stress (for a given impactor/indenter geometry and rock condition) that must be exceeded to produce fractures in a brittle target.

To increase understanding of impact phenomena on rock, the concepts of coefficient of restitution, rock fracture due to impact, and penetration are discussed in this section.

3.3.1 Coefficient of Restitution

One way of describing a collision of two bodies (such as a cutter and a semi-infinite rock mass) is by the coefficient of restitution. This coefficient, e , is measured using a free-falling impactor and a motionless, semi-infinite, horizontal target:

$$e = v_f / v_i \text{ or } (h_f / h_i)^{1/2} \quad (3-1)$$

where:

v_f = initial rebound velocity of an impactor

v_i = impact velocity of an impactor

h_f = rebound height of an impactor

h_i = initial drop height of an impactor

The coefficient of restitution defines the degree of elasticity of an impact, and the transfer of impact energy between colliding bodies. For a perfectly plastic collision, $e = 0$, and for a perfectly elastic collision, $e = 1$. For energy to be conserved, plastic impact results in the permanent deformation of the target and/or the impactor. Since in the case of a tungsten carbide button impactor on a concrete or rock target, it can be assumed that negligible plastic deformation of the impactor occurs, and that all of the deformation takes place in the target.

One study that was reviewed [Inyang and Pitt, 1991] investigates the relationship between the restitution of an impactor and the permanence of deformation of a target rock. In this study, the transfer of kinetic energy from an impactor to a rock target was investigated. E_o is defined as the kinetic energy of a cutter just before impact, and is equal to the sum of the energy used in the deformation of the rock, E_u , and the elastic rebound energy, E_r . Thus, the relation:

$$E_o = E_u + E_r \quad (3-2)$$

The ratio of E_r/E_o is defined as the energy transfer coefficient, K_1 . It can be shown that:

$$K_1 = 1 - (h_f/h_i) = 1 - e^2 \quad (3-3)$$

At low energy impact, K_1 is very small, representing an elastic collision. Once the critical impact energy (the impact energy required to cause fractures) is exceeded, K_1 approaches 1, representing a plastic

collision, with most of the kinetic energy being absorbed in the formation of cracks in the target. K_1 may be useful for evaluating the efficiency of the low velocity/high mass vs. high velocity/low mass impact.

Other pertinent work involving the coefficient of restitution was done by Radil et al. [Radil et al., 1994]. In this study, the effects of impact velocity on the coefficient of restitution were investigated by dropping steel balls on a target plate at impact velocities ranging from about 2.7 to 4.3 feet per second. The time intervals between successive impacts were measured to calculate the coefficient of restitution for each impact. It was found that as impact velocity increased, the coefficient of restitution decreased. This result has implications with regard to impact velocity effects of the RAMEX cutter. Based on Inyang and Pitt's work [Inyang and Pitt, 1991] relating the energy transfer coefficient to coefficient of restitution (Equation 3-3), it can be seen that K_1 increases with an increase in the coefficient of restitution. This implies that since the coefficient of restitution increases with impact velocity, more plastic deformation (i.e. fracturing) is expected at higher impact velocities, and a higher proportion of initial impactor energy is imparted to the target.

3.3.2 Rock Fracture Due to Impact

Inyang and Pitt's study [Inyang and Pitt, 1991] also analyzed the elastic to plastic transition of rock deformation due to impact. Typically, impact deformation of rock is both elastic and plastic, the proportions of which depend on the magnitude of impact energy and the physical properties of the rock. In the case of low energy impact, some elastic deformation of the rock may occur with little or no plastic deformation. As impact energy is increased, a point is reached (the critical impact energy) where the rock is deformed beyond its elastic limit, and permanent deformation occurs due to crushing and fracturing. Critical impact energy is a function of target strength (tensile and compressive), brittleness, distribution of target material flaws, and target porosity. It is also a function of impactor geometry. If impact loads are applied at energies below the critical impact energy, the process may be considered "inefficient" because fractures will not form to cause chips. Conversely, excessive impact energy leads to excessive crushing and localized cracking, which also may result in "inefficient" pulverization of the rock.

Another study by Inyang and Pitt [Inyang and Pitt, 1990] provided additional insights into the subject of rock fracture due to impact. In that study, they present results of an investigation into optimum indenter spacing for rock excavation systems that use impulse loads. A non-destructive testing method to measure specimen sonic wave velocity was developed for estimating the critical impact energy of rock, and also the

size of the fracture zone around individual impact points. The impacts were imposed on rock specimens at various impact energies using a rock fracture hammer (Figure 3-3) designed specifically for the tests [Inyang and Pitt, 1993]. The indenter on the rock fracture hammer was nearly identical in shape to those used on the RAMEX cutter. The rocks tested included Lac Du Bonnet Granite, having a UCS of 26,000 psi, and Academy Black Granite, having a UCS of 29,000 psi.

The tests were performed in two stages. First, the specimens were impacted with the rock fracture hammer at increasing impact energy magnitudes until the sonic wave velocity in the specimen showed a significant reduction due to the formation of cracks. Each specimen was impacted only once during the tests. The critical impact energy of the specimens was defined as the minimum impact energy at which a significant reduction in the sonic wave velocity occurred. A representative plot of percent decrease in sonic wave transit time versus impact energy from these tests is shown in Figure 3-4. The critical impact energy for the Lac Du Bonnet Granite was about 29 ft-lb, and that for the Academy Black Granite was about 37 ft-lb. Since the indenter used in these tests was of similar geometry to that of the RAMEX cutter indentors, the reported critical impact energies of the granite specimens should correlate well with the critical impact energies (per indenter) that would be observed using the RAMEX cutter on similar specimens.

The second stage of Inyang and Pitt's tests [Inyang and Pitt, 1990] involved delineating the size of the fractured zone caused by impacting the specimen at its fracture energy. This was done by drawing a fine grid on the specimen, and measuring the sonic wave velocity along the grid after impact. The results indicated a fracture zone approximately 5.5 to 7.0 in. diameter. Visual observation of the cracks using dye penetrant confirmed these results. The results of their study suggest that impact points on mechanical excavation tools should be spaced to take advantage of the size of the fracture zone created during impact. The objective of impact bit spacing is to slightly overlap fracture zones so that cracks interact from adjacent impacts (Figure 3-5). If spacing is too large, additional force will be necessary to remove fragments because the cracks cannot interact. If impacts are too close, some impact energy will be wasted. For the rocks tested in the Inyang and Pitt study, a bit spacing of 5 to 6 in. was recommended.

FIGURE 3-3 - Rock Fracture Hammer
 (Source: Inyang and Pitt, 1993)

Figure 3-4 - Percent Decrease in Wave Velocity vs. Impact Energy for Lac Du Bonnet Granite
(Source: Inyang and Pitt, 1990)

FIGURE 3-5- Impact Spacing Model for Subsurface Cracks in Brittle Rock
(Source: Inyang & Pitt, 1990)

Many impact loading experiments on concrete specimens have been performed by other investigators [Green, 1958; Bentur et al., 1986; Mindess et al., 1985]. A study by Bentur et al. [Bentur et al. 1986] investigated impacts on concrete beams using a drop-weight apparatus similar to the apparatus used in the present study. While the target in Bentur's work was different from that used in this study, several interesting points were made regarding the testing equipment. The most significant was that large amounts of impact energy may be lost to the testing machine, depending on its structural rigidity. Several impact studies [Bentur et al., 1986; Mindess et al., 1985] have been performed using a particular drop testing apparatus located at the University of British Columbia. While features of that testing device were not incorporated into the present study, future research efforts may investigate some of its beneficial features to improve results of future experiments.

3.3.3 Penetration Due to Impact

Fuchs [1963] presented an analytical study of impact in which he developed a semi-analytical-based predictor equation for projectile penetration into a semi-elastic target. An similar type of equation is found in technical reports by Young [Young, 1967; Young, 1972]. It is important to note that the studies by both Fuchs and Young were for projectiles impacting targets at velocities much greater than the velocity associated with the RAMEX or HEHH cutters, but still well below the sonic velocity of the rock. Also, the predictor equations were developed for projectiles that penetrate a distance of at least three times the diameter of the projectile. Young [Young, 1967; Young, 1972] reports that a lower limit on the applicability of the predictor equations exists for penetrations of lesser depths, because the mechanisms of penetration may not be the same. An analytical solution to shallow penetration, or cratering by impact, has not been developed, but the aforementioned penetration predictor equations may be used as a first order estimate [Young, 1967].

Fuchs' equation was developed assuming that the instantaneous force of resistance on an impactor is a function of impact velocity, $R(v)$. The equation can be written as:

$$R(v) = f_1(v^0) + f_2(v^1) + f_3(v^2) \quad (3-4)$$

In this equation, the first phase of projectile penetration (at high velocity) is governed mainly by resistive forces proportional to v^2 . At moderate velocity (during deceleration), the resistive forces are proportional to v , and in the final phase of deceleration, they are mainly static.

The first term, $f_1(v^0)$ is defined as the product:

$$f_1(v^0) = aH = \alpha \quad (3-4a)$$

where:

a = cross sectional area of the projectile

H = unidentified constant related to target hardness (or static resistance)

Note: In Chapter 5, an attempt to approximate H in terms of Schmidt Hardness is presented.

The third term, $f_3(v^2)$, is defined as :

$$f_3(v^2) = Ca(\gamma/2g)v^2 = \beta \quad (3-4b)$$

where:

C = projectile shape factor (equal to $2/3$ for a hemispherical projectile)

a = cross sectional area of the projectile

γ = density of the target material

v = velocity of the projectile at the point of impact

g = acceleration due to gravity.

The middle term, $f_2(v^1)$, which describes the component of resistive force that governs the motion of the projectile during deceleration is defined as:

$$f_2(v^1) = 2\alpha^{1/2}\beta^{1/2}v \quad (3-4c)$$

Thus the original expression for resistive force can be rewritten as:

$$R(v) = \alpha + 2\alpha^{1/2}\beta^{1/2}v + \beta v^2 = (\alpha^{1/2} + \beta^{1/2}v)^2 \quad (3-5)$$

Equation 3-5 suggests that resistive force on a penetrating projectile is a function of the impact velocity, projectile geometry, target density, and target hardness.

To derive the penetration predictor equation, Equation (3-5) can be rewritten as:

$$R(v) = -m(dv/dt) = -m(d^2x/dt^2) = (\alpha^{1/2} + \beta^{1/2}v)^2 \quad (3-6)$$

where:

m = impactor mass.

x = penetration

Separation, substitution and integration of this equation leads to an expression for penetration, x , as a function of mass and impact velocity:

$$x(v) = \frac{m}{\beta} \left\{ \ln \left(1 + \frac{\sqrt{\beta}}{\sqrt{\alpha}} v \right) - \frac{\sqrt{\beta} v}{(\sqrt{\alpha} + \sqrt{\beta} v)} \right\} \quad (3-7)$$

Another equation was developed by Young [Young, 1967; Young, 1972] for estimating penetration depth, P from measurements of impacts on soil and rock targets. This equation is a function of projectile mass, velocity, and projectile cross-sectional area, and includes three empirically derived constants. Young's equation is as follows:

$$P = 2KSN \sqrt{\frac{W}{A}} \ln(1 + 2v^2 \times 10^{-4}) \quad (3-8)$$

where:

K = mass correction factor

S = dimensionless soil constant

N = nose performance coefficient

W = impactor mass (kg)

A = impactor cross-sectional area m^2

v = impactor velocity (m/sec)

In Equation 3-8, the soil constant, S , is the most subjective parameter and has a broad range of values based target material properties. In Chapter 5, an attempt will be made to relate S to the Schmidt Hardness value.

CHAPTER 4 METHODS OF INVESTIGATION

This chapter describes the testing methods and experimental procedures used for the present study. Laboratory testing involved drop tests on concrete and rock specimens. Many tasks were necessary to prepare for these tests, including concrete specimen preparation and strength testing, preliminary drop testing, and test plan organization. Methods for the reduction of data were also developed so that the experiments could be evaluated effectively.

The drop tests involved constructing a laboratory version of the RAMEX cutting tool, attaching it to a vertical sliding mechanism on the drop test apparatus (Figure 1-5), then dropping the cutter onto test specimens. The drop test apparatus was located at the U.S. Bureau of Mines (USBM) Pittsburgh Research Center (PRC). Drop height and cutter mass were varied to achieve different velocity impacts at a *constant impact energy*. Data were collected during the tests to evaluate whether impact velocity significantly affects rock breakage.

Initial tasks conducted prior to drop testing included:

- development of concrete mixtures to satisfy the requirements of the test specimens
- preparation of concrete test specimens
- non-destructive strength testing of concrete specimens (dynamic and Schmidt Hammer testing)
- direct concrete strength testing (UCS and Brazilian tensile strength) to assure that the specimens met the pre-imposed requirements
- design of a specimen containment box ("rock box")
- design of a laboratory impact cutter for drop tests
- preliminary drop tests using a simplified test apparatus
- development of a test plan for the formal drop tests

Details of the specimen preparation, concrete mix design and concrete testing are found in Appendix A.

4.1 LABORATORY TEST SPECIMEN DESIGN

Although the RAMEX cutter is primarily designed for use in hard rock applications, concrete test specimens were selected for the present tests. The reasons behind this decision were primarily practical. Foremost, considering the successful past performance of the RAMEX system during field testing on 30,000+ psi rock [Handewith et al., 1989, 1990], there was no question of its ability to effectively cut

strong material. Since the objective of the present experiments was to study the influence of *velocity* on cutter performance in order to evaluate the possibility of using the RAMEX cutter on a HEHH-type delivery system, the strength of a specimen was not the most important test parameter. Instead, ensuring similar physical properties to "typical" rock including fracture behavior, brittleness, and tensile to compressive strength ratio was considered more important. Similarities between the physical properties of concrete mixtures and many rocks types have been observed by other investigators [Goldsmith, 1960, Green, 1958].

Other factors such as the consistency of the concrete specimens were considered. A limited number of laboratory tests were performed to ensure this consistency and thus minimize the number of variables in the experiments. Since concrete is easily and inexpensively reproduced with a high degree of consistency [PCA, 1990], the use of concrete also allowed the test specimens to be consistent with each other. Rock specimens, on the other hand, often vary in physical characteristics over short distances, which could lead to large variations in test results.

4.1.1 Concrete Specimens

Several requirements of the concrete mix were established early in the project. The most important physical characteristics to control were the strength, aggregate gradation, setting time, and shrinkage characteristics. The desired characteristics of the concrete are summarized in Table 4-1.

Two different concrete mixes were used in the tests. These had unconfined compressive strengths (UCS) that differed by a ratio between 2 to 3. Other than strength, the physical characteristics of the concrete specimens were kept relatively constant between specimens to reduce the number of test variables. Three concrete test specimens of each strength (low and high) were prepared. Details of the concrete mix designs, specimen preparation, and strength testing are presented in Appendix A.

The dimensions of the test specimens were selected to allow up to five rows of drop tests at spacings between 2 and 4 in. This range of spacing was selected based on the preliminary field tests with the RAMEX mining boom [Handewith et al., 1989, 1990]. The specimens also were required to be sufficiently massive to withstand the impact energy of the tests without completely breaking, but small enough to easily fit in the testing apparatus. The final specimen dimensions were 22 by 14.5 in., and 8 in.

thick. Each specimen weighed about 200 lb. Figure 4-1 is a photograph of a test specimen being placed in the specimen containment box.

TABLE 4-1 - Concrete Test Specimen Desired Properties

PROPERTY	CHARACTERISTIC	REASON
Aging/Curing	Rapid	To expedite specimen preparation
Strength	Two very different unconfined compressive strengths: (2 to 3 times difference)	To evaluate effects of varied specimen strength on impact velocity effects and button row spacing
Maximum Aggregate Size	< 1/4-in.	To reduce effects of crack meandering during tests
Aggregate Grain Size Distribution	Well graded	To provide specimen homogeneity, strength, and reduced shrinkage
Shrinkage Potential	Minimal	To allow specimen to be constrained during tests
Reproducibility of Specimen	Simple	To allow easy replication of future specimens

The thickness was selected with the hope that cracks would not propagate through the specimen during testing, so that both sides of the specimen could be used. No preliminary tests were conducted to evaluate whether this thickness was sufficient to withstand impact without through-cracking; unfortunately, all specimens experienced through-cracking during testing.

4.1.2 Specimen Containment Boxes

Each specimen was contained in a steel containment box approximately 1 in. larger in all plan dimensions than the specimen. Gypsum cement (Hydrostone™; United States Gypsum Company, 1995), which is a quick-hardening, slightly expansive, plaster-like cement with high strength (approximately 10,000 psi), was placed between the concrete specimen and the containment box (Figure 4-1).

The specimen containment boxes were designed with the intent to provide lateral constraint of the test specimens to simulate a confined condition. A confined condition was desired to preclude the through-cracking of the specimens during testing. The side walls were constructed with C8 x 11.5 channel steel

stock, bolted in-place. The bases of the boxes was constructed of 0.50 in. steel plate stock, bolted to the channel stock side walls.

FIGURE 4-1 - Representative Concrete Test Specimen

4.2 TEST CUTTER DESIGN

The 6 in. long test cutter was fitted with three hemispherical tipped, tungsten carbide button indentors, installed at 3 in. centers along a single row (Figure 1-5). This single row configuration was used so that the spacing between impact rows could be varied. The buttons were of the same design and dimensions (0.75 in. diameter by 1 in. long) as those utilized by the prototype RAMEX cutter (Figure 1-1). The buttons were recessed 0.74 in. into a AISI 1018 steel holder, leaving 0.26 in. of button exposed.

4.3 PRELIMINARY DROP TESTS

To better understand what could be expected to occur in the drop tests, and to assist in the development of a test plan for the PRC tests, preliminary drop tests were conducted at the University of Washington in the summer of 1995. These tests were conducted using a 39 lb cylindrical steel rod (approximately 3 in. diameter by 22 in. long) as an impactor. One tungsten carbide button indenter was installed at the end of this impactor. Tests were conducted by raising the rod to a pre-determined height with a cable, then dropping it through a vertical guide tube onto concrete test specimens. The drop height was selected to duplicate the impact energy of 250 ft-lb per button, which was used in the drop tests at PRC. Qualitative observations of concrete durability, fracture patterns, crater generation, and instrumentation response were made in the preliminary tests. Manual data collection methods were rehearsed to establish standard methods. Also, an accelerometer/oscilloscope set-up was used to evaluate the length of the impact event. This provided information for advance determination of instrument set-up configurations at PRC, including data sampling rate.

The preliminary tests aided in developing a test plan for the PRC tests, and to observe how the test specimens would withstand impact testing. No formal results were recorded from these preliminary tests, but significant observations were made during the tests, including:

- Through-cracking of preliminary test specimens, suggesting that through-cracking of the PRC test specimens was a strong possibility.
- Careful mating between the test specimens and the bottom of the containment box was important to minimize stress concentrations and bending moments which could lead to failure of the specimens by through-cracking.
- A peripheral "no-impact zone", of 3 in. or more from all edges of the specimen was necessary to prevent spalling of the specimen along the edge on impact.

- Craters could be volumetrically measured by filling the resulting crater with glycerin.
- The impact event, (i.e. vibratory response of specimens from impact) was between 0.3 and 0.7 seconds in length.

4.4 DROP TESTS

After the preliminary testing was complete, the drop tests at the PRC were performed in late summer, 1995. The drop tests were performed on the concrete test specimens described above, and on two different types of natural rock specimens: Berea Sandstone and Barre Granite.

4.4.1 Drop Testing Program

The drop tests were divided into two groups: row dimensioning tests and production tests. The testing program was developed to organize the row dimensioning tests so they could be completed in the four day time period in which the testing laboratory and equipment were available. The objectives of the row dimensioning tests were to establish the optimum row spacing for the production tests, and to aid in the refinement of the RAMEX cutting tool design. The production tests used a constant spacing of impact rows, and were performed in September and October, 1995 by the USBM. The impacting patterns for both sets of tests are illustrated in Figure 4-2.

DROP TEST METHODS

For all tests, a "conditioning" row of impacts was made approximately 3 in. from the specimen edge. The first row spacing was then measured from this conditioning row. The 3 in. no-impact zone around the perimeter of the specimens unfortunately proved to be insufficient to prevent spalling and through-cracking of the test specimens.

The impact "indexing" (as opposed to row spacing) was defined as the spacing between the center of the impact buttons *along a row*. The buttons on the test cutter were spaced on 3 in. centers. After the first drop on an impact row, the cutter was "indexed" 1 in. along the row, and then dropped a second time. A

third drop was accomplished after another indexing of 1 in. The resulting completed row consisted of nine individual impacts, each 1 in. apart.

When a row of impacts had been completed, one or more repeat impacts were imposed on that row. This was done to produce definitive interacting fractures between rows, and to simulate the repeated impacts of the RAMEX system to some degree. Repeat impacts were made until the button penetration was deep enough so that the cutter body contacted the specimen. If chips had not formed by the time this occurred, no further impacts were made.

a) Row Dimensioning Tests

b) Production Tests

Test Specimen Plan Dimensions: 14.5 in. by 22 in. (all dimensions in inches)

Figure 4-2 - Row Dimensioning and Production Test Impact Layouts

When tests were completed on a specimen, the specimen was removed from the steel containment box, and the condition of the specimen was observed. In general, all specimens through-cracked in multiple locations due to the impacts.

DATA COLLECTION

Data collection involved both digital and manual data recording. Digital data recording utilized the instrumentation described below. Data were also collected manually by removing the fragments produced from the tests, weighing them, photographing the specimen, and measuring crater volumes with either glycerin or modeling clay.

Digital Data Recording

Data recorded during tests were stored electronically on floppy disks in voltage units. Conversions from voltage to engineering units using the scaling factors in Table B-2 (Appendix B) were performed using a BASIC program written for that purpose. The converted data files were used in the analysis of the tests, as described in a later section.

Digitally collected data for each impact included:

- Time
- Cutter displacement
- Normal and side cutter forces
- Acceleration (cutter assembly)
- Acceleration (specimen box)

Approximate cutter velocity at impact was evaluated at the time of testing with a data recording software package. This software package evaluated the change of cutter displacement (Δx) over a user-specified time interval (Δt). It was noted in the initial tests that the velocities thus calculated were significantly lower than expected for free fall. As a consequence, the impact energy imposed on the specimens was slightly less than originally calculated. This reduced velocity was attributed to friction in the cutter guide rails and to the method of analysis used by the data recording software, which calculated slightly lower impact velocities than the actual value. Trial drops were conducted to determine the necessary drop height to accomplish the desired impact velocity, but exact measurement of the velocity was not possible

during testing because of deficiencies in the data recording software. The software did, however, enable the ability to assure that the drop velocities were different by a factor of two while delivering a nearly consistent impact energy. Achieving the value of 250 ft-lb impact energy was not as important to as maintaining the relative velocity difference between the low and high velocity tests.

Manually Collected Data

Much information was obtained during the drop testing that could not be digitally recorded. This information was manually collected once a row of impacts was completed, or in some cases, after fragmentation was observed between rows. The manually collected data included:

- Photographs of impacts
- Assessment of chip formation
- Fragment collection
- Crater volume by clay imprint and glycerin volume measurement

The fragments generated during impact were contained on the specimen surface using a guard (with an open top for the cutter) placed on top of the specimen. After completing an impact row, the fragments were collected by hand and placed in plastic bags. It was recognized that all of the fragments could not be collected this way, particularly the dust-size particles. Furthermore, impact-induced fractures did not always result in loose fragments and some had to be removed from the specimen with assistance. For this reason, fragments were considered "loose" only if they could be easily removed by hand without the use of additional tools.

Crater volumes were measured at selected times, typically after a row of impacts resulted in chip formation. These measurements were obtained by filling the craters with glycerin until the fluid surface was flush with the undamaged surface of the specimen. The volume of the crater was recorded as the volume of glycerin added to the crater. These measurements were not made on craters where through-cracks were present.

Clay imprints of craters on selected specimen surfaces were also collected using synthetic modeling clay. The clay was pressed into the crater and then heated to form a hardened, archival record of the disturbed surface. These imprints were also used to qualitatively evaluate cutter penetration for the different specimen strength/impact velocity combinations.

4.4.2 Testing Apparatus

The testing apparatus consisted of the impact delivery system, and a data recording system. The following sections provide a general description of the testing apparatus components. The setup of the testing apparatus was performed independently by the US Bureau of Mines (USBM); details of the testing apparatus will be provided in a forthcoming report by the USBM.

IMPACT DELIVERY SYSTEM

As described earlier, the drop tests were performed at a constant impact energy, but at different impact velocities. Due to limitations of the testing equipment, the rock specimens were tested at low velocity impact only. The target impact energy on the specimens was approximately 250 ft-lb per button indenter. This impact energy was selected based on the design impact energy per button indenter of the RAMEX delivery system, and on typical impact energy ratings of hydraulic breakers.

The impact delivery system derived its impact energy from gravity, using a nearly free-falling mass to which the test cutter was attached. The system consisted of a main carriage and a counter balance mounted on adjacent vertical tracks, and connected with steel cable (Figure 1-5). Test specimens were mounted and clamped in a fixture at the base of the apparatus. Hydraulically operated pistons and worm gears were used to position each specimen for impact.

The impact velocity of the *main* carriage was controlled by adding or subtracting weights from the counterbalance. The maximum impact velocity of the main carriage alone could be adjusted from 0.4 - 11 ft/sec. A *cutter* carriage was mounted on the *main* carriage; low friction, linear motion bearings were used to facilitate cutter carriage travel along two parallel vertical guide rails, which were part of the main carriage. The test cutter was mounted on the bottom of the *cutter* carriage. A mechanically operated, quick release hook released the *cutter* carriage, causing the cutter to drop and impact the specimen. The impact velocity of the cutter was controlled by the height from which the *cutter* carriage was dropped. The cutter and cutter carriage are shown in detail in Figure 4-3.

Figure 4-3- Test Cutter and Cutter Carriage

For the low velocity (approximately 10 ft/sec) tests, the cutter carriage was raised and dropped without the use of the main carriage. For the high velocity (approximately 20 ft/sec) tests, both the main carriage and cutter carriage were raised and dropped. The main carriage supplied an initial velocity before the mechanical hook released the cutter carriage into free-fall. The initial velocity was required to achieve the desired impact velocity for the high velocity tests.

The impact energy of approximately 250 ft-lb per button required 750 ft-lb of total energy at the cutter just before contact with the test specimen. Using basic kinematic principles, this impact energy would

theoretically (i.e. neglecting friction and air resistance) be achieved when cutter mass and drop height were as follows:

High velocity (20 ft/sec)/low mass (130 lb) tests: drop height = 74.6 in => impact energy = 750 ft-lb
Low velocity (10 ft/sec)/high mass (485 lb) tests: drop height = 18.6 in => impact energy = 750 ft-lb

Unfortunately, friction in the system reduced the apparent acceleration due to gravity, and therefore the impact velocity. The apparent acceleration due to gravity, factoring in the effects of friction and air resistance, was calculated as 31.38 ft/sec/sec rather than 32.2 ft/sec/sec using the displacement-time history record (Appendix B). Based on this, the actual tests were performed with the cutting tool mass and drop height as follows:

High velocity (20 ft/sec)/low mass (130 lb) tests: drop height = 78.0 in => impact energy = 720 ft-lb
Low velocity (10 ft/sec)/high mass (485 lb) tests: drop height = 19.0 in => impact energy = 651 ft-lb

INSTRUMENTATION

The data acquisition system consisted of a 16 channel, 12 bit converter, and programmable gain amplifiers with a maximum sampling rate of 62,500 samples/sec. Maximum test length at this acquisition rate was 32.768 ms. For the first half of the testing program, only 8 channels were used with a sampling rate of 250 samples/sec, resulting in maximum test acquisition time of 1.8 sec. For the second half of the testing program, a finer response resolution was used and the sampling time was reduced, increasing the sampling rate to 16,666 samples/sec. Instruments to measure force, acceleration and cutter displacement were assigned to data acquisition channels. The data acquisition channel assignments and conversion factors from voltage to engineering units are presented in Appendix B, Table B-1.

One of the force channels was used to trigger data acquisition upon impact. The instruments were then able to recall "pre-trigger" data points from memory, providing data prior to impact. The force dynamometer consisted of four piezo-electric force rings clamped between two steel plates, and mounted within the cutter shaft. Force was measured in three orthogonal directions; normal and tangent to the specimen surface (i.e. perpendicular and parallel to the impact rows). The forces could be measured to $\pm 80,000$ lb.

The cutter accelerometer was installed in the force dynamometer assembly. The accelerometer measured shock and vibration up to 500 g's with a frequency range of 0.7 to 10,000 Hz. Another accelerometer was

magnetically mounted on the specimen constraining box to monitor vibration of the sample. This accelerometer operated in the range of ± 100 g's with a response of .7 to 6,000 Hz. Unfortunately, the box accelerometer was not operational (out of range) for many of the tests.

Cutter displacement was initially measured with a low friction slide potentiometer, mounted on one of the guide rails of the cutter carriage. Displacement was therefore measured only when the cutter was moving along the rails. A calibration curve was established by the USBM relating voltage output from the slide potentiometer to cutter displacement. Unfortunately, the slide potentiometer failed midway through the testing program, and had to be replaced with a string potentiometer.

4.4.3 Data Reduction

The digitally recorded data for each impact event was summarized in terms of time-history records of cutter displacement, cutter forces, and cutter acceleration to facilitate analysis. Since there is too much data to present herein, only sample data sets representing different velocity/specimen strength test variations are presented in Appendix B for comparison (i.e., low velocity impact/low strength specimen, low velocity impact/high strength specimen, etc.). A directory is provided at the beginning of Appendix B to guide the reader to the data sets.

Each data set consist of time-history plots of normal cutter force, side cutter force, cutter acceleration (vibration), box acceleration (vibration), and cutter displacement vs. time. Details of these plots are also shown for the main impact event, to provide better resolution of this event. Also included in each data set is a comparison of the measured cutter displacement during free-fall with a displacement function based on Newton's gravitational equations. A plot of the residual between the data and calculated values of displacement is also presented. A detailed plot of cutter displacement just prior to impact for each data set is also shown, with a super-imposed best fit curve that was used to determine impact velocity. Finally, a velocity-time history determined using numerical differentiation is provided for each data set.

Data sets for most impacts recorded in the testing program were analyzed to determine characteristics associated with each cutter impact. Table 4-3 summarizes the parameters analyzed for all data sets. Details of the methods of analysis for the impact data are presented below.

TABLE 4-3 - Calculated Parameters from Data Reduction

Parameter	Determined by:
peak force in cutter (side and normal)	inspection of force-time history
impulse (area under force-time history curve)	integration of force-time history
impulse duration	inspection of force-time history
average normal force in cutter	impulse/pulse duration
peak acceleration (cutter and box)	inspection of force-time history
cutter rebound time	inspection of force-time history
impact velocity	inspection of displacement-time history
impact energy	calculation ($\frac{1}{2}mv_i^2$)
coefficient of restitution	(rebound height/ drop height) ^{1/2}
cutter penetration	$\frac{1}{2}(v_i \cdot \text{impulse duration})$
net specific energy	impact energy/removed material

Note: v_i = impact velocity

Peak values of force, and acceleration due to the impact tests were determined from the respective time histories. Cutter rebound time (time between the main impact event and the rebound impact), and impulse duration (duration of main impact event) and were determined from inspection of normal force-time histories. Penetration time (time from initial main impact to peak normal force) was determined by inspection of the detailed force-time histories. According to Goldsmith [Goldsmith, 1960], plastic deformation (i.e. penetration) is assumed to cease when the contact force between an impactor and target attains its maximum value. However, some experimental results have indicated some plastic deformation occurring subsequent to the peak contact force. A conservative calculation of cutter penetration will result from assuming that maximum penetration occurs at the time of peak contact force.

Impulse was calculated by numerically integrating the normal force-time history, i.e.:

$$I = \int_{t_o}^{t_f} F(t) dt \quad (4-1)$$

where:

I = impulse, lb-sec
 F = normal force, lb
 t_o = time at beginning of impact, sec
 t_f = time at end of impact, sec

Average normal cutter force was defined as the impulse divided by the impulse duration:

$$F_{avg} = I(t_f - t_o)^{-1} \quad (4-2)$$

The cutter velocity at impact was calculated by plotting the displacement-time history just prior to impact, and determining the slope of the best fit line to the data. To plot the displacement-time history, the displacement transducer had to be calibrated to convert voltage to engineering units. This was done using a polynomial regression fit. Impact velocity determined from the best-fit line is referred to as the "slope method". An example of the slope method for each data set in Appendix B is provided.

The slope method gave results which were generally supported by an second method for velocity determination where the displacement-time history was differentiated numerically, resulting in a velocity-time history. A velocity-time history for each data set in Appendix B is provided therein. From the velocity-time history, impact velocity was determined by manually drawing a best fit curve to the displacement-time history, then reading the velocity corresponding to the time of impact. The numerical differentiation method gave varied results; in addition, there was considerable scatter near the point of impact. Because of this, little confidence was placed in the results. This method did, however, provide reassurance to the results obtained using the slope method.

As a third approach to the velocity measurement, a fit of the displacement-time history curve was established based on Newton's equation of displacement for a free-falling object (Equation 4-3). Examples of these curve-fits are provided for each data set in Appendix B. In this method, the form of the equation of free-fall motion from rest was used to establish a best fit curve to the displacement-time data (Equation 4-3).

$$s = \frac{1}{2}at^2 \quad (4-3)$$

where:

s = displacement, ft
 a = vertical acceleration, ft/sec/sec
 t = time since release, sec

The recorded displacement-time history was plotted, and super-imposed with Equation 4-3. The acceleration was varied until a minimal residual (i.e., the difference between the measured displacement and the curve model) was attained. The value of acceleration that resulted in the minimum average residual for all data sets was taken as the "apparent" acceleration due to gravity, corrected for the friction

of the cutter carriage and air resistance. The apparent acceleration due to gravity for these experiments (a^*) was determined as:

$$a^* = 31.38 \text{ ft/sec/sec} \quad (4-4)$$

Once the impact velocity was established for both the low and high velocity tests, the impact energy could be determined. For ease of calculation, an average impact velocity for all data sets was used for the low and high velocity tests to determine impact energy. Impact energy was calculated as:

$$E = \frac{1}{2}mv_i^2 \quad (4-5)$$

where:

E = impact energy, ft-lb

m = mass of cutter, slugs

v_i = average impact velocity, ft/sec

The coefficient of restitution, e , for each impact was calculated from the cutter rebound time using fundamental dynamics principles:

$$e = v_2/v_1 = (h_2/h_1)^{1/2} \quad (4-6)$$

where:

v_1 = impact velocity

v_2 = initial rebound velocity

h_1 = drop height

h_2 = rebound height = $1/8(gt^2)$ [Radil et al., 1994]

(t = rebound time, $g = a^* = 31.81 \text{ ft/sec/sec}$ to account for friction)

The coefficient of restitution is used in Chapter 5 for evaluation of energy transfer at the different impact velocities.

One of the most important parameters for analyses in this study was the depth of cutter penetration into the specimen. This was calculated from the penetration time (Δt) and impact velocity. The penetration time (Δt) is the time from the instant of cutter contact on the specimen to the time corresponding to maximum normal cutter force (i.e. the time of maximum cutter penetration [Goldsmith, 1960]).

Assuming that the deceleration of the cutter is linear from the time it impacts the specimen to the point at which the cutter reverses direction, the average velocity of the cutter between impact and arrest can be taken as:

$$v_{avg} = \frac{1}{2} (v_i) \quad (4-7)$$

and cutter penetration into the specimen. P is:

$$P = v_{avg} (\Delta t) \quad (4-8)$$

where:

Δt = impulse duration (determined by observation from force-time history)

P = penetration

In conjunction with data reduction, additional tests were conducted on the collected fragments of material generated during the tests. These tests included:

- grain-size distribution analysis
- volume measurements of material removed by impacts

Grain size analyses using ASTM D421 were performed on selected fragments collected during the row dimensioning tests. This was done to provide a qualitative evaluation of the gradation of fragmented material produced during the tests. The complete results of these tests are too lengthy to include herein, but general results are discussed in Chapter 5.

The volume of material removed per row of impacts was measured using Archimedes principle. The fragments were added to a known volume of water, and the displacement due to the fragments was taken as the volume of material removed. The volume of fragments removed per button per blow was estimated by dividing the volume of the fragments collected by the number of buttons and the number of impacts.

Specific energy (Equations 2-1 and 2-2) was calculated by determining the energy consumed during fragmentation of one row, and dividing this energy by the mass of the fragments generated. Specific energy was also determined in terms of crater volume as measured by two methods. In the first method, the volume of the crater was determined from glycerin measurements. In the second, the volume was calculated from the fragments collected, using on the Archimedes method of volumetric measurement.

CHAPTER 5 RESULTS AND DISCUSSION

5.1 - TEST SPECIMEN PROPERTIES

The concrete test specimens utilized in the experiments are described in Appendix A. Two rock specimens, Berea Sandstone and Barre Granite, were also tested. The test specimens represent a broad range of physical properties. Table 5-1 summarizes average values of some important physical specimen properties as determined in this study and by others [Kretch et al., 1974]. The range of values for the physical properties of the concrete specimens is presented in Tables A-3 and A-4 (Appendix A).

TABLE 5-1 - Summary of Physical Properties of Test Specimens

Specimen	UCS ⁽¹⁾ (psi)	ITS ⁽¹⁾ (psi)	UCS to ITS Ratio	Dynamic Young's Modulus (psi)	Density (pcf)	Schmidt Hardness ⁽²⁾ (R)
Low Strength Concrete ⁽³⁾	3,620	360	10.1	3,000,000	135	39
High Strength Concrete ⁽³⁾	9,915	572	17.3	5,000,000	142	52
Berea Sandstone ⁽⁴⁾	6,700	155	43.2	2,000,000	132	52
Barre Granite ⁽⁴⁾	28,600	1,110	25.8	6,600,000	165	70

⁽¹⁾ Reported values for concrete specimens are average of all tests, based on 28 day cure time

⁽²⁾ Tested with Type-I hammer

⁽³⁾ Reported values for saturated concrete specimens are estimated, based on ASTM C-215 method

⁽⁴⁾ Source: Kretch et al., 1974

The values reported for dynamic Young's modulus of the concrete specimens were not calculated using the same methods to those used with the rock [Kretch et al., 1974]. Consequently, the dynamic modulus values for the concrete specimens should only be compared to each other, and not to the values reported for the rock specimens. The same is true for the reported values of density. Since the density was calculated for saturated concrete specimens, while the water content of the rock specimens was unknown. Generally, however, the porosity of the rock is very low, and the water content adds little to its density.

With the exception of Berea Sandstone, the UCS to ITS ratios of the test specimens increase with increasing UCS. Higher UCS to ITS ratios coupled with higher UCS values generally indicates more

brittle behavior. This was demonstrated by the more violent failure of the high strength concrete specimens during UCS testing (Appendix A). This more violent failure may also be attributed to more strain energy in the testing apparatus at failure. The Schmidt hardness values roughly correlate with both UCS and density, as observed by other researchers [Atkinson, 1993].

5.2 - CUTTER VELOCITY/ENERGY CALCULATIONS

Table C-1 (Appendix C) presents the results of velocity calculations for individual impact events, based on two different velocity calculation methods: the "slope" method and numerical differentiation (Chapter 4). The average, maximum, and standard deviation for all tests are also presented. A summary of the average impact velocities are presented in Table 5-2. The average velocity values for the concrete specimens were 9.3 and 18.9 ft/sec for the low and high velocity tests, respectively. These are close to the target velocities of 10 and 20 ft/sec. The average calculated velocity for the rock specimen tests was 9.8 ft/sec, which is slightly higher than the average low velocity for the concrete specimens. This difference is probably related to variations in frictional resistance on the cutter slide, since the starting positions were different for the thicker rock specimens than for the concrete. Although the impact velocities determined by the two methods were in good agreement, the "slope method" was selected as the working method for calculating impact energy because it was believed to be more precise.

TABLE 5-2 - Summary of Average Impact Velocities* and Impact Energy

	Low Velocity ft/sec	High Velocity ft/sec	Ratio of High to Low Velocity	Impact Energy per Button: Low Velocity Tests (ft-lb)	Impact Energy per Button: High Velocity Tests (ft-lb)
Concrete	9.3	18.9	2.03	217	240
Rock	9.8	NA	NA	241	NA

* Velocities calculated by Slope Method

NA= Not Applicable

The impact energy for the concrete tests, as determined from the calculated impact velocity and cutter mass, was close to the target impact energy of 250 ft-lb per button indenter. For the concrete tests, the calculated energies for the low velocity impacts were approximately 13% below the target energy, and 9% below the impact energies for the high speed tests. Although the difference in impact energy is small

between the low and high velocity tests, it may have contributed to the higher productivity of the high velocity tests, as discussed below.

5.3 GENERAL TEST RESULTS

5.3.1 Visual Observations

Visual observations of specimen reaction to the impacts were made during the drop test experiments. Most significantly, all concrete specimens through-cracked at some point during testing (Figure 5-1). This problem was not observed in the rock specimens, which were nearly three times thicker than the concrete specimens. This suggests that the constraining system for the concrete specimens was not sufficiently stiff to properly constrain the specimens. This through-cracking may have significantly influenced some of the results in the study.

FIGURE 5-1 - Impact Rows on Concrete Test Specimen with Through-Cracks

In many of the tests where impact rows were spaced at greater than 2 in., chip formation was not observed between the rows. This is in contrast to the field tests of the prototype RAMEX mining boom, which produced chips at spacings of 3 in. [Handewith, 1989], and suggests that the laboratory testing did not replicate field conditions. The lack of chip formation in the drop tests may be partially due to loss of energy to through-cracking. This is illustrated in Figure 5-1 where 3 in. spaced impact rows on a high strength concrete specimen resulted in no chips formation, and through-cracks are clearly observed in the bottom impact row. Chip formation in the field tests may also be attributed to rapid frequency impacts at a much higher *total* impact energy than in the laboratory tests.

Deeper cratering of the specimen surface was qualitatively observed to be associated with through-cracks along the impact rows. The impact rows with through-cracks, for example, were observed to have craters that were deeper and "V"-shaped (e.g. Figure 5-1, bottom row) than the craters in rows without through-cracks (e.g. Figure 5-1, top rows).

While it would have been desirable to analyze only data from specimens without through-cracks, this was not possible because all specimens through-cracked at some time during testing, particularly on the conditioning row. Once cracked, the specimens probably exhibited greater energy attenuation properties, which are believed to have affected the test results, especially the values of cutter forces, and cutter and box acceleration. The calculated parameters of impulse, coefficient of restitution and energy transfer coefficient (K_1) could also have been affected. The through-cracked specimens likely behaved more like a highly fractured rock mass than an intact one, and for this reason, the results of the tests may not be representative of intact rock. The results, nevertheless, can be used to compare the relative differences between the effects of the test variables on the specimens.

Figures 5-2 and 5-3 illustrate post-testing results on low strength concrete specimens where chips formed. Figure 5-2 shows a low strength specimen that has been impacted two times at 3 in. spacing, prior to removal of the chips and fragments. Figure 5-3 shows a similar test specimen after chip and fragment removal. In specimens where interacting fractures formed chips, the fractures developed at a very low angle to the specimen surface, resulting in a shallow crater. Generally, the crater depths were approximately the same as the calculated cutter penetration depth in Figure 5-3.

FIGURE 5-2 - Chip Formation in Low Strength Concrete Specimens, Prior to Chip Removal

FIGURE 5-3 - Chip Formation in Low Strength Concrete Specimens, After Chip Removal

Figure 5-4 illustrates post-testing results on a high strength concrete specimen without chip formation. This specimen was impacted four times at 3 in. spacing. Note the linear indentation along the sides of impacts marking where the cutter body "bottomed out" on the specimen. Also, on the bottom row of impacts, through-cracking occurred. This resulted in deeper cutter penetration in that row than on the top row, where through-cracking did not occur.

Figure 5-5 illustrates the results of tests on Barre Granite. Note that the three sets of impacts at 2-in. spacing in the foreground (bottom of photo) resulted in minor specimen damage compared to the concrete specimens. The singular impact location in the back (top of photo) is the result of numerous impacts without translation of the specimen. No chips were formed in the Barre Granite during testing. In the case of Berea Sandstone, however, chips were formed consistently throughout testing. This is believed to be due to thin laminations in the rock parallel to the impact surface, which may have represented planes of weakness. The chips in the Berea Sandstone were similar in appearance to those formed in the low strength concrete specimens (Figure 5-1).

FIGURE 5-4 - Post-Impact View of High Strength Concrete Specimen; No Chip Formation

FIGURE 5-5 - Post-Impact View of Barre Granite; No Chip Formation

5.3.2 Sieve Analyses

The fragment size distribution due to impact was assessed by sieve analyses. Fragments generated by the cutter included all specimen cuttings, including the special case where chips were formed by interacting fractures between impact rows. The sieve analyses indicates that the fragments are generally well-graded when chips are not generated, with all fragments being smaller than 1.5 in. When chips formed, fragment distribution was gap-graded. No chips larger than 4 in., however, were observed. Problems were encountered in the grain size distribution tests because many of the chips were delicate and broke during testing. This resulted in a higher percent of fines being observed than actually formed during the tests. This effect may have been off-set to some degree from the inability to collect all fines during the drop tests.

5.4 - IMPACT CHARACTERISTICS

Impact characteristics for individual impacts are presented in Table C-2 (Appendix C), along with average values and the standard deviation for all tests. Average values are summarized in Table 5-3, and are illustrated in Figure 5-6a-d.

TABLE 5-3 - Impact Characteristics Summary

Velocity/Specimen	Maximum Impact Force, lb			Impulse lb-sec	Maximum Acceleration (cutter), g's	Maximum Acceleration (box), g's	Rebound Time, sec	Coefficient of Restitution	Energy Transfer Coefficient
	Normal	(L to R*)	(cut)						
Low Velocity / Low Strength	35,726	1,986	3,105	44	174	54	0.188	0.29	0.91
Low Velocity / High Strength	45,180	3,471	3,285	44	495	68	0.178	0.28	0.92
Low Velocity / Berea Sandstone	51,391	2,362	3,496	48	708	70	0.186	0.29	0.91
Low Velocity / Barre Granite	69,908	4,139	4,246	51	912	147	0.177	0.28	0.92
High Velocity / Low Strength	35,982	1,589	1,797	17	453	---	0.211	0.16	0.97
High Velocity / High Strength	38,409	2,494	2,927	16	608	97	0.190	0.15	0.98

(L to R) = Cutter forces perpendicular to impact row direction

(cut) = Cutter forces parallel to impact row direction

--- indicates that no reliable data were recorded for this event

5.4.1 - Velocity Effects

Based on the test results for the specimens, average maximum cutter forces (both normal and side) do not appear to be significantly influenced by velocity (Figure 5-6a), although normal forces are slightly higher in low velocity tests than in high velocity tests. These results may be important as design input when considering the use of the RAMEX cutter on impact delivery systems with different impact velocities.

The results also suggest that at a constant impact energy, impulse magnitude is significantly influenced by impact velocity (Figure 5-6b). For specimens of equal strength, low velocity impulse (measured in the cutter) was approximately 2.5 times greater than high velocity impulse, indicating that there is an inverse correlation between these variables. While this may suggest predicted differences in cutter performance at different impact velocities, no such effect is supported by the measurements of damage to the specimens. Field testing of the cutter at different impact velocities is recommended to further evaluate the effects of cutter impulse on performance.

FIGURE 5-6a - Cutter Forces

FIGURE 5-6b -Impulse Magnitude

It is also possible that the higher impulse for low velocity tests was due to the relatively flexible connection of the weights on the cutter. During the tests, these weights were observed to bounce at impact, which could have resulted in longer duration impacts. In the high velocity tests, no weights were attached to the cutter, and this bouncing effect was not observed. Since impulse is calculated as the integral of the cutter force-time history, longer duration impacts would result in higher impulse. Because the average cutter force and impact (damping) time is a function of this impulse, the higher magnitude impulse at low velocity suggests a higher *average* force on the specimen:

$$F_{avg} = I/\Delta t^* \quad (5-1)$$

where:

F_{avg} = average impact force
 I = impulse magnitude
 Δt^* = impact (damping) time

The maximum vibrational acceleration of the cutter at low velocity was approximately 40 to 80% of high velocity cutter acceleration on comparable strength specimens (Figure 5-6c). Thus the maximum vibrational acceleration directly correlates with impact velocity. Unfortunately, the frequency content of the vibrational acceleration was not determined. Analysis of the maximum vibrational acceleration coupled with the frequency content may be an interesting direction for future research concerning cutter wear.

The maximum vibrational acceleration measured on the specimen constraint box, as shown in Figure 5-6c, was not reliably recorded. This is because the recording instrument (accelerometer) was often not in place during testing, and when used, its useful range of 100 g's was often exceeded. For this reason, no box acceleration measurements were recorded during the high velocity/low strength tests due to either equipment malfunction or out-of-range data. Based on the data that was collected, the maximum box acceleration appears to be slightly higher for high velocity impact tests than for low velocity impact tests on comparable specimens.

The rebound time, or time between the main impact and rebound impact, was also compared for different impact velocities on comparable specimens. The rebound height, as determined from rebound time (Chapter 4), does not appear to be influenced by impact velocity (Figure 5-6d). This suggests that the coefficient of restitution (Chapter 3) is affected by impact velocity, and is much lower for the high velocity tests. This is because, for a constant coefficient of restitution, the rebound height is the same percent of the initial drop height, regardless of the mass of the cutter. This result is in agreement with other studies [Radil et al., 1994] that suggest that the coefficient of restitution is dependent on impact velocity.

FIGURE 5-6c - Vibrational Acceleration: Cutter and Rock Box

FIGURE 5-6d - Coefficient of Restitution and Energy Transfer Coefficient

The coefficient of restitution for high velocity tests is about 50% lower than for low velocity tests on comparable specimens, indicating an inverse relationship between the two. This also indicates that the energy transfer coefficient, K_1 (Chapter 3) is higher for the high velocity impacts. Since K_1 represents the fraction of initial kinetic impact energy consumed by plastic rock deformation, the test results suggest that more plastic deformation results from a high velocity impact, under constant impact energy. The nature of this plastic deformation, however, is not clear since no differences in crater size were observed and the nature of subsurface cracks was not investigated.

5.4.2 - Specimen Strength Effects

The maximum normal cutter force generally increased with specimen strength (Figure 5-6a). This effect is more noticeable for the low velocity tests than for high velocity tests. Side cutter forces on the other hand appear to be random (Table C-2), and do not appear to be significantly affected by either velocity or specimen strength. The side forces may be attributed to specimen heterogeneity and to unevenness of the contact between the cutter and specimen. For example, if an oversize aggregate is encountered, the cutter may be forced to the side, resulting in a side force. Similarly, if the cutter encounters an undulation on the specimen surface at impact, bending moments and side forces will result.

Impulse magnitude (and therefore impact duration and average cutter force) does not appear to be significantly affected by specimen strength for similar impact velocities (Figure 5-6b). While the average impulse magnitude for the rock specimens appears to be slightly higher than for the concrete specimens, this may be due to the slightly higher impact velocity for the rock specimen tests.

Maximum cutter acceleration generally increased with specimen strength (Figure 5-6c). This is as expected, since impacts on higher strength specimens are generally more elastic than with lower strength specimens, which show more plastic failure and lower elastic moduli.

The coefficient of restitution and energy transfer coefficient were not significantly affected by specimen strength (Figure 5-6d). Although the coefficient of restitution between an impactor and target has been observed to increase with increasing target strength (e.g. Goldsmith, 1960) this was not observed in this study. One possible explanation is that the cutter impacted the specimens above the "critical impact energy", which is the energy required to cause fractures [Inyang and Pitt, 1991]. As the impact energy increases above the critical impact energy, the amount of plastic deformation also increases. Because of

this, the impact energy is significantly greater than the critical impact energy, the coefficient of restitution for different specimens may approach a similar value. Other investigators [Inyang and Pitt, 1991] tested the critical impact energy of several granite specimens, and found the fracture energy to be between 30 to 40 ft-lb using a hemispherical impactor similar to the ones in this study. This is much lower than the impact energy per button used in this study (approximately 250 ft-lb). In addition, the critical impact energy of the concrete specimens and of Berea Sandstone is probably much lower than that of granite.

5.5 - INSTANTANEOUS CUTTER PENETRATION

5.5.1 Penetration Based on Measured Crater Depths

Cutter penetration was estimated both manually, and by back-calculation from the penetration-time data. The manual measurements of penetration included both profilometer readings and modeling clay imprints. These measurements do not reflect instantaneous penetration because each specimen was impacted repeatedly at the same location before these measurements were made. Because of this, the measurements are of limited value, and should be used for comparison purposes only. Table 5-4 summarizes these data. It is important to note that each measurement represents only one event, and therefore a statistical evaluation could not be made.

TABLE 5-4 - Measured Crater Depths

		Maximum Crater Depth Measured from Clay Imprints				Maximum Crater Depth Measured from Profilometer Profiles		
		1-Impact	2-Impact	3-Impacts	4-Impacts	2-Impacts	3-Impacts	4-Impacts
Low Velocity	Low Strength	0.35	0.49	--	--	0.80	--	--
	High Strength	--	0.30	--	0.57	--	--	0.60
	Berea Sandstone	--	--	--	--	0.68	--	--
	Barre Granite	--	--	0.24	--	--	0.25	--
High Velocity	Low Strength	--	--	--	--	0.80	--	--
	High Strength	--	0.54	--	--	0.55	--	--

The crater depths for two impacts on the high strength specimen suggests that high velocity impact penetrated the specimen deeper than low velocity impact. In contrast, profilometer measurements on low strength specimens for two impacts show no measurable differences in penetration at different velocities. The crater depth data in Table 5-4 also suggest that the depth of cutter penetration decreases with specimen strength. The measurements in Table 5-4 (two impact, clay imprint) can be regarded as an upper bound on the calculated instantaneous cutter penetrations, since they generally represent multiple impacts, and the calculated cutter penetration represent only one impact. It is of interest to note that the measurement made after a single low velocity impact on a low strength specimen shows a penetration close to the calculated instantaneous penetration value for that case, as discussed below.

5.5.2 Calculated Cutter Penetration

Calculated penetrations for individual impacts are presented in Tables C-4 and 5-5. These are based on the time of penetration, assuming a linear decrease in velocity from the time of impact to the time of maximum penetration, as determined from the force-time histories (Chapter 4).

**TABLE 5-5 Summary of Observed Cutter Penetration
Compared with Analytically Predicted Penetration**

Velocity/Specimen	Calculated Penetration in.	Predicted Penetration, in. (Fuchs*)	Predicted Penetration, in. (Young*)
Low Velocity / Low Strength	0.22	0.22	0.20
Low Velocity / High Strength	0.14	0.16	0.15
Low Velocity / Berea Sandstone	0.14	0.18	0.16
Low Velocity / Barre Granite	0.10	0.13	0.10
High Velocity / Low Strength	0.22	0.24	0.18
High Velocity / High Strength	0.20	0.18	0.14

* modified

In general, the calculated penetrations (Table 5-5) decrease with specimen strength, as expected. The calculated penetration, however, does not appear to be significantly affected by impact velocity. The one exception is the high strength concrete specimen, where the average calculated penetration is about 0.06 in. (about 25%) greater for the high velocity impact than the low velocity impact. This difference may partially be due to the 9% greater impact energy in the high velocity tests. Further study should be

conducted using several different impact velocities to verify this conclusion.

FIGURE 5-7 - Cutter Penetration

Given that cutter penetration may one of be the best laboratory indicators of the effect of cutter impact velocity, a second method to verify penetration calculations such as high speed photography would be useful, but was not possible in this study . Unfortunately, the importance of cutter penetration as a measurement was not recognized until after the laboratory testing program was complete, and alternate measurements for penetration were not made. It is important to note that, while the calculation of the cutter penetration using the method described in Chapter 4 is recognized as a good approximation [Goldsmith, 1960], its accuracy may have been compromised by assuming that the cutter velocity decreases linearly between impact to cutter arrest. Non-linear cutter deceleration may be a source of error in these calculations. Future research should focus on better and more accurate ways to measure cutter penetration, as well as methods of separating elastic and plastic deformations during cutter penetration.

5.5.3 Penetration Prediction Estimates

As discussed in Chapter 3, two penetration predictor equations (Equations 3-7 and 3-8) were used for comparison with calculated penetrations. Both predictor equations [Fuchs, 1963 and Young, 1972] use general terms to describe the physical properties of the target. An attempt was made to replace the hardness term (H) in Fuchs predictor equation with an equivalent function using the Schmidt hardness

value (R). This term was determined by iteration assuming that H is a multiple of R. The resulting equation for H is :

$$H = (100,000)R \quad (5-3)$$

where:

H = Fuchs equation target hardness factor

R = Schmidt hardness number (Type L hammer)

In Young's equation, the soil term (S) was also determined by iteration as a function of the Schmidt hardness value, i.e.:

$$S = 0.02(100-R) \quad (5-4)$$

where:

S = Young's equation target "soil term"

R = Schmidt hardness number (Type L hammer)

The predicted penetrations using the modified Fuchs' and Young's equations are compared to the calculated instantaneous penetrations in Table 5-5 and are illustrated by the bar graph in Figure 5-7. As can be seen, these are close to the calculated penetration values in most cases. Further study, however, is required to demonstrate the use of the modified predictor equations for other types of target specimens.

5.6 - SPECIFIC ENERGY CALCULATIONS

Table C-5 (Appendix C) presents specific energy calculations, including specific energy values calculated from the weight and volume of chips generated for each impact row. Table 5-6 summarizes specific energy results in terms of both weight and volume for various impact row spacings and velocity/strength combinations. Two specific energy results based on direct volume measurements are presented: one calculated from the crater volume as measured with glycerin and the other from measurement of the volume of water displaced by the cuttings. Of the three specific energy measurements, the specific energy values based on weight are believed to be the most accurate since the volume measurements were highly variable.

TABLE 5-6 - Summary of Calculated Specific Energy

Velocity/Specimen	Row Spacing, in.	Specific Energy hp-hr/ton	Specific Energy hp-hr/cu yd (1)	Specific Energy hp-hr/cu yd (2)
Low Velocity / Low Strength	0	11.4	10.5	--
	2	1.8	1.7	3.7
	3	0.7	0.9	1.2
	3	1.9	1.0	--
	4	5.9	--	--
Low Velocity / High Strength	0	13.7	--	26.7
	2	2.4	3.7	4.9
	3	8.3	15.1	--
	3	7.2	--	14.2
	4	7.5	--	--
Low Velocity / Berea SS	3	2.5	3.6	--
Low Velocity / Barre Granite	0	26.2	24.6	53.7
	2	21.3	26.2	--
	3	24.5	39.9	--
High Velocity / Low Strength	3	1.6	1.9	2.8
	3	1.1	0.8	--
	4	5.4	--	--
High Velocity / High Strength	2	1.9	--	3.5
	3	1.5	1.1	2.6
	3.5	1.6	1.1	4.1
	4	5.1	--	--

(1) Crater volume determined using glycerin

(2) Volume determined by Archimedes Principle

Generally, lower specific energies were observed for low strength concrete and rock specimens at all impact velocities. The values ranged from 0.7 hp-hr/ton for low strength/low velocity tests, to 24.5 hp-hr/ton for Barre Granite/low velocity tests (all at 3-in. spacing). For the low strength specimens, specific energy was not significantly different between low and high velocity impact tests. However, the high strength/high velocity impact tests showed a much lower specific energy than for the high strength/ low velocity tests because inter-row fragments only formed in the high velocity tests. This result agrees with the higher penetration observed in the high velocity tests on the high strength concrete. It is not clear if this would be the case in the field, where vibration effects due to rapid impacts could cause fragments to be loosened more readily. In addition, the slightly higher impact energy in the high velocity tests may account for the chips forming in these tests, and not the low velocity tests.

Due to the small amount of specific energy data collected, no statistical basis exists for assessing the performance of the cutter at different velocities using specific energy. A field study is therefore recommended since larger quantities of material could be generated resulting in more accurate specific energy measurements.

5.6.1 Row Dimensioning Tests

One of the purposes of performing row dimensioning tests was to determine an "optimum" cutter spacing for the specimens, based on minimum specific energy. A plot of specific energy vs. spacing is presented in Figure 5-8. This indicates that optimum spacing for the low strength concrete was between 3 and 3.5 in. and approximately 2 in. for high strength concrete and Barre Granite. Berea Sandstone was tested at spacings of 3 in. only, but based on the low value of specific energy at that spacing (2.5 hp-hr/ton), it is likely that 3 in. is close to optimum.

FIGURE 5-8 - Specific Energy vs. Impact Row Spacing

CHAPTER 6

SUMMARY, CONCLUSIONS AND RECOMMENDATIONS

The objective of this research was to determine if any differences exist between low impact velocity/high cutter mass and high impact velocity/low cutter mass impacts on the effectiveness of the RAMEX hard rock cutting tool. The impact velocity/cutter mass combinations were adjusted to deliver a constant impact energy. This was done to evaluate whether the RAMEX cutter would perform comparably on two different impact delivery systems which utilize different impact velocities and cutter masses for impact. These systems include the RAMEX mining boom and the conventional hydraulic hammer.

To accomplish this objective, a series of laboratory tests were performed in which a test cutter similar to the RAMEX cutter was dropped on specimens of concrete and rock using a drop test fixture located at the U.S. Bureau of Mines Pittsburgh Research Center. The test specimens consisted of low and high strength mortar (approximately 3000 and 10,000 psi, respectively), Barre Granite and Berea Sandstone.

The measurements used to evaluate the effectiveness of the cutter under the different impact velocity/cutter mass combinations included instantaneous cutter penetration, specific energy, cutter force, cutter vibration, impulse, and coefficient of restitution. Based on the test results, it was concluded that instantaneous cutter penetration was the best laboratory measurement for identifying the effectiveness of the cutter. The effect of impact row spacing on cutter efficiency was also tested.

The results of the tests showed no significant differences in the depth of instantaneous cutter penetration under the different impact velocity/cutter mass combinations. A number of general trends were identified from the other measurements that could be pursued in future research efforts. These include:

- Impulse magnitude and coefficient of restitution were primarily a function of impact velocity. Normal cutter force at impact, on the other hand, was primarily a function of rock strength. The cutter side forces were variable and are believed to be due to specimen heterogeneity and uneven surface contact of the cutter. Cutter acceleration was a function of both impact velocity and rock strength.

- Specific energy was not significantly affected by variations of impact velocity and cutter mass, given a constant impact energy. Specific energy is believed to be a less reliable indicator of cutter performance under laboratory conditions than instantaneous penetration because of the difficulties of accurately measuring the quantity of the excavated material.
- The optimum impact row spacing for chip formation ranged from 2 to 3.5 in., based on specific energy calculations. The optimum row spacing was approximately 3 in. for weaker materials, and 2 in. or less for high strength materials. These results may not correlate well with field tests, where repeated impacts may increase the amount of rock fragmentation.

The results of the tests suggest that the impact velocity/cutter mass variations do not significantly affect the cutter performance. Based on this, both low velocity/high mass and high velocity/low mass delivery systems can be expected to deliver similar performance results in the field. The results of the specific energy calculations support this conclusion.

During the tests, a number of problems were encountered with the testing apparatus and data collection system. Because of time constraints, the laboratory tests were performed over a period of only four days, so these problems were responsible for a significant loss of data that may have otherwise been collected. As a result, there was a lack of sufficient data for the statistical analysis of some of the test results. Many of these problems would have been difficult to anticipate given the lack of previous research in this area, however, they may provide useful insights for the design of similar testing programs in the future.

The significant problems encountered with the testing apparatus and data collection system in these tests include:

- The controls for the winches used to position the cutter were difficult to fine-adjust to obtain a repeatable drop height. As a result, impact velocity was difficult to control. Fine adjustment of the drop height was time consuming, and the repeated adjustments finally resulted in the failure of the winch by overheating.
- To obtain the high velocity impact, complicated operation of the test apparatus was necessary. Because of these operations, high velocity tests were unable to be performed on the rock specimens.

It is recommended that any future test apparatus be mechanically simplified and have the ability to easily achieve the desired impact velocities with adequate reproducibility. Drop testing devices used by others (e.g. Mindess et al. 1985) should be investigated for ideas to improve upon the design.

- The test apparatus had the ability to attach weights to the cutter for the different impact velocity/cutter mass combinations. However, the weights were not rigidly attached and moved noticeably during the impact event. This may have affected impulse and coefficient of restitution measurements. Also, the cutter carriage should have a "dynamic brake" so the cutter does not bounce after the main impact event, which supplies additional impact energy to the specimen that was neglected in this study.
- The data recording system was not scaled correctly for some of the measurements, and periodically became inoperable. As a result, some data was out of range or not recorded. Future tests should include preliminary tests to assure that the data collection instruments are scaled to the proper range that will be experienced in the tests.

Another significant problem encountered in the tests concerned the test specimen confinement boxes. The test specimens developed through-cracks because of insufficient confinement during testing. Because of this, measurements of maximum cutter force, cutter acceleration, impulse, and the coefficient of restitution may have been affected. Additionally, chip formation between cutter paths may have been severely reduced from what might have occurred if through-cracking had not occurred. Instantaneous cutter penetration was calculated from impacts where through-cracks had not occurred, therefore, it is believed that the through-cracks did not significantly effect the cutter penetration results. Future studies should utilize a stiffer specimen box to provide more lateral confinement of the specimens. A circular specimen containment system is recommended to provide stiffer confinement of the specimen since lateral restraint would be supplied by tension in the frame (hoop stresses) rather than bending resistance.

Some general recommendations for future research projects in this area include:

- Future laboratory tests directed specifically toward instantaneous cutter penetration should use only one tungsten carbide indenter to eliminate the effects of specimen surface irregularities that may distribute impact energy unevenly among indentors. The design of such a test cutter should be such that the button holder does not "bottom-out" on the specimen after repeated impacts. Additionally more than two velocity/mass combinations (at constant impact energy) should be tested to clarify the relationship between impact velocity and cutter penetration.

- Field tests comparing the impact velocity/cutter mass combinations should be conducted to verify the results of the laboratory tests. Such tests should be evaluated by comparing specific energy of the different velocity/mass systems.
- Experiments to determine optimum row spacing should be performed in the field. To determine optimum row spacing, cutters of varied row spacings should be used, and performance compared using specific energy. Several cutter spacings (i.e. four or more) could be used to better define the relationship between row spacing and specific energy.
- A system for specifying impact energy should be established for the RAMEX cutter, based on the rock properties. For example, a broad range of HEHH ratings (i.e. impact energies) exist that could be used, depending on the competence of the rock to be excavated. Future research efforts could incorporate field tests of the RAMEX cutter using a range of impact energies on a single rock type to establish its "optimum" impact energy (based on specific energy). The critical impact energy for the test rock could then be determined with a device similar to the rock fracture hammer (Chapter 3). This process could be repeated for several rock types of different strengths to establish a clear relationship between optimum impact energy and critical impact energy. This relationship could then be used by designers to specify the appropriately rated impact delivery system for different rock types, based on their critical impact energy.

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APPENDIX A

CONCRETE TEST SPECIMEN PREPARATION AND STRENGTH TESTING

A.1 CONCRETE MIX DESIGN AND SPECIMEN PREPARATION

Concrete mix designs were developed in this study to produce test specimens that satisfied the guidelines of Table 4-1 (Chapter 4). The two target unconfined compressive strengths (UCS) for the concrete mixes were 3000 psi (low strength) and 10,000 psi (high strength). Aggregate requirements specified a maximum aggregate size smaller than a U.S. No. 4 sieve, classifying the mix as a mortar. Also specified were properties such as low shrinkage and rapid curing time.

Mix designs for the low strength mortar were commonly available and easy to produce. The high strength mortar was unusual (i.e., a "high performance" mix), and a recipe for such a mix was not commonly available. Being specialized, high performance mix designs are typically proprietary. As such, a custom mix design for the high strength test specimens was developed, and trial batches were prepared at the University of Washington materials laboratory to evaluate the physical properties of the mix. For completeness, trial batches of the low strength mix were produced as well. The evaluation of specimen mix designs included aggregate gradation tests, and a series of conventional strength tests on cured test specimens. The results of these tests were used to evaluate whether the concrete mixtures were appropriate for use as test specimens.

A fundamental parameter of the mix design is the water to cement ratio (w/c). The w/c is the controlling factor for many of the important qualities of concrete, and is usually the "starting point" for designing a concrete mix. Using a low w/c generally results in a higher strength concrete (Figure A-1). In addition, properties such as shrinkage potential and workability are effected by w/c. The minimum w/c for complete hydration of Portland cement is 0.22 to 0.25 (by weight). A w/c of 0.4 (by weight) produces a relatively strong, low shrinkage, workable mix. A w/c below 0.4 typically results in a very strong mix with low workability, unless admixtures are used.

Type III cement was selected for use in developing both the high strength and low strength mixes due to its high early strength and rapid drying rates. The high early strength of Type III cement results from its extremely fine aggregate size.

Admixtures affect the performance of the cement paste, and can provide properties that are not typical of conventional pastes. Two admixtures were used in the high strength mix: silica fume and water reducer. No admixtures were used in the low strength mix. Silica fume is a by-product of the electronics industry and is composed primarily of silicate-based materials. Its microscopic size and cementitious properties

add to the strength of a concrete by in-filling and cementing very small voids in the concrete. Being spherical in shape, silica fume also increases workability of the mix when wet, resulting in a smaller water requirement for the mix and higher strength and less shrinkage of the cured concrete. Water reducer is an admixture that increases the workability of an otherwise stiff concrete without the addition of more water. Workability is generally a function of the stiffness of the mix, which in turn is governed by the water content and w/c. Increased workability also results in less segregation of the concrete during placement, resulting in a more homogeneous mixture and minimal weakening due to "surface bleeding".

FIGURE A-1 - General Relationship Between Unconfined Compressive Strength (UCS) and Water/Cement Ratio in Concrete
(Source, PCA, 1990)

The standard test for evaluating the stiffness of the mix is the slump test (ASTM C143). For the high strength concrete, slumps in the range of 2 to 3 in. were observed, indicating a low to moderately workable mix. Slumps for the low strength mix ranged from 6 to 8 in., indicating a highly workable mix.

Aggregate is the primary ingredient of concrete, and its properties strongly affect the performance of the mix design. Aggregate was limited in size to that passing the No. 4 sieve in order to minimize its

influence on crack propagation and chip formation during impact testing [Mihashi et. al., 1992]. The aggregate used for the test specimens was also selected to avoid shrinkage complications associated with certain gradations, mineralogies and particle shapes [PCA, 1990]. ASTM C33 sets forth gradation specifications for fine aggregate. Table A-1 and Figure A-2 show the well-graded aggregate distribution that was used in the mix design for the test specimens, compared to the tolerable gradation range from ASTM C33.

TABLE A-1 - Aggregate Gradation Data

US Sieve Size	Sieve Opening (mm)	Cumulative Weight Retained (grams) (Specimen)	Cumulative Percent Retained by Weight (Specimen)	Cumulative Percent Passing by Weight (Specimen)	Allowable Cumulative Percent Passing by Weight (ASTM C-33)
3/8 in	9.52	0.0	0.00	100.00	100
3	6.68	0.0	0.00	100.00	98 to 100
4	4.76	17.5	1.25	98.75	95 to 100
8	2.36	217.8	15.60	84.40	80 to 100
20	0.841	540.2	38.70	61.30	45 to 80
40	0.420	993.3	71.14	28.86	15 to 45
100	0.149	1359.4	97.36	2.64	2 to 10
200	0.074	1387.5	99.41	0.59	0 to 5
pan	0.000	1395.8	100.00	0.00	—

FIGURE A-2, Grain Size Distribution of Aggregate

In the development of the concrete mixtures, the target water content for the mix designs was selected as 320 lb/cu yd. based on the characteristics of the selected aggregate. The aggregate water content was determined prior to batching, and moisture adjustments to the saturated, surface dry (SSD) condition were made to the mix design using standard methods [PCA, 1990]. The cement content was then determined for each of the mixes (low and high strength), based on this water content and target w/c. The target w/c for the mixes was 0.80 for the low strength and 0.30 for the high strength mixes. The remainder of the mix (aggregate content and admixtures) was designed around the target w/c, using the PCA guidelines [PCA, 1990]. The two concrete mortar recipes (low and high strength) used for the test specimens are presented in Table A-2.

TABLE A-2 - Concrete Mix Designs for Test Specimens

(a) LOW STRENGTH CONCRETE MIX

Material	For 1 CU. FT.		For 1 CU. YD.	
	lb	Absolute Volume, cu ft	lb	Absolute Volume, cu ft
water	15.70	0.25	423.9	6.8
cement (Type III)	19.50	0.10	526.5	2.7
fine aggregate (SSD)*	104.50	0.62	2821.5	16.9
air (1.5% by volume)	0.00	0.02	0.0	0.5

(b) HIGH STRENGTH CONCRETE MIX

Material	For 1 CU. FT.		For 1 CU. YD.	
	lb	Absolute Volume, cu ft	lb	Absolute Volume, cu ft
water (not incl. SF sol'n H ₂ O)	10.30	0.17	278.1	4.5
cement (Type III)	33.00	0.17	891.0	4.5
fine aggregate (SSD)*	104.00	0.62	2808.0	16.8
silica fume solution (H ₂ O + solids)	3.63	0.04	98.01	1.14
HRWR** (ml)	—	325	—	8775
air (1.5% by Volume)	—	0.015	—	0.41

*SSD = Saturated, surface dry aggregate

**HRWR = High Range Water Reducer

To minimize segregation and settlement of aggregate due to excessive working, the wet concrete was placed within the forms in horizontal layers of relatively uniform thickness. After placement, air was removed from the concrete by rodding and tapping the forms with a hammer. Surface finishing was performed by first screeding with a straight edge when the concrete was wet, then floating when the surface bleed water had evaporated. Curing of the test specimens was performed in a moist environment

by covering the specimens with wet burlap and plastic sheeting at an ambient temperature of 50-70° F. The specimens were kept in this condition for a period of 7 days. This minimized shrinkage and the development of internal tensile stresses due to rapid loss of water during curing.

Concrete test cylinders (4 in. diameter by 8 in. long) were prepared for unconfined compressive strength (UCS) tests and indirect tensile strength (ITS) tests. After casting, the test cylinders were cured for 14 and 28 days in a humid environment (70-100% relative humidity) and at a constant temperature of approximately 70°F. Cylinders were prepared for UCS testing by capping with sulfur according to ASTM C39. Test cylinders were made for UCS testing from an initial trial batch and the subsequent test specimen batch. A separate batch was made for preparing ITS test cylinders. All test cylinders for strength testing were made using the same mix proportions of Table A-2.

A.2 CONCRETE STRENGTH TESTING

All strength testing of concrete specimens was conducted at the University of Washington materials laboratory. UCS of concrete test cylinders was determined using the procedures given in ASTM C39. In this test, the test cylinder is compressed at a controlled rate of loading until failure. The UCS is then calculated as the load at failure divided by the cross-sectional area of the test cylinder. The testing procedures for ITS tests involved loading test cylinders on their sides at a controlled rate according to ASTM C496, resulting in a splitting failure. The tensile strength of the test cylinder can be approximated from the force at failure as:

$$T = 2P/\pi dL \quad (A-1)$$

where:

- T = splitting tensile strength (psi)
- P = maximum applied load (lb)
- d = average test cylinder diameter (in)
- L = average test cylinder length (in)

This method, sometimes referred to as the Brazilian Splitting Tensile Strength test, is reported to give a good approximation of true tensile strength of the test cylinder [Brook, 1993].

Brittleness was evaluated qualitatively by observation during the strength testing of all test cylinders, according to a definition by Williams [1942]. A brittle failure is such that broken fragments resulting

from an imposed load on a body can be fitted together to the exact same shape as before failure load was applied: brittle failure often occurs in a catastrophic manner, with fragments being rapidly ejected and accompanied with a sharp noise.

Non-destructive testing measurements in this study included estimation of the dynamic Young's Modulus and UCS using a dynamic test method, and the measurement of specimen hardness using the Schmidt hammer hardness test.

The dynamic tests were performed 2 to 3 times per week for 4 weeks after mixing. The dynamic test specimens were composed of 3 by 4 by 16 in. concrete mortar beams, prepared from the trial batch using the mix proportions in Table A-2. Two beams were prepared for each (high and low strength) mix design. The primary purpose of the dynamic tests was to give an early indication of whether the trial batches met the compressive strength ratio specification of Table 4-1. As explained below, the non-destructive tests did not yield accurate, absolute measurements of the specimen's physical properties. The goal of these tests was to compare the physical characteristics, particularly UCS and dynamic Young's modulus, of the low and high strength mixes relative to each other. As a result, there is little confidence in the absolute values of dynamic Young's modulus and UCS from the dynamic tests, but the ratios between these properties are probably reliable.

The dynamic tests were performed according to ASTM C215. In these tests, the fundamental resonant frequencies of the test beams were determined using an impact resonance method. Each beam specimen was supported as a double cantilever on two taut steel wires. The specimens were then struck with a small impactor, and the responses were measured with a piezoelectric accelerometer and recorded with an oscilloscope. Fundamental frequencies of the beam were determined using a Fourier transformation of the acceleration-time history. The dynamic Young's modulus, E_d , of the concrete was determined by:

$$E_d = CWn^2 \quad (A-2)$$

where:

W = weight of specimen, (lb)

n = fundamental transverse frequency of specimen, (Hz)

C = $0.00245(L^3T/bt^3)$

L = length of specimen, (16 in.)

t, b = dimensions of cross section of beam, in. (t = 4 in., b = 3 in.)

T = correction factor, (1.40)

The correction factor, T , is based on an estimate of the Poisson ratio for the concrete, which was estimated. The same correction factor was used for all specimens so their estimated dynamic modulus could be compared relative to one another. Accurate determination of dynamic modulus with this method would require an accurate measurement of the Poisson ratio.

The dynamic Young's modulus is proportional to the square root of the UCS of the concrete. An approximate relation between the dynamic Young's modulus, E_d , and UCS is [PCA, 1990]:

$$E_d = 57,000(UCS)^{1/2} \quad (A-3)$$

The UCS can also be estimated by many indirect tests, including the Schmidt hammer test. The Schmidt hammer measures concrete hardness by impacting the surface with a special hammer, and measuring its rebound height. While the hardness is an important physical property of its own [Williams, 1942; Atkinson, 1993; Brook, 1993; Nelson, 1993], correlations of UCS versus Schmidt hardness have also been made. The UCS estimated from the Schmidt hardness in the present study, however, did not correlate well with the results of direct UCS tests and are not reported herein. However, the raw Schmidt hardness values are reported in the next section.

Schmidt hammer testing was conducted using a Type-L Schmidt hammer on the cured test specimens. Each test was conducted by impacting the specimen with the hammer seven times. The two lowest test values were eliminated, and the remaining five values were averaged, resulting in an average Schmidt "R" value. Data were collected by varying the locations of the seven impacts, and by testing in the same location for all impacts. The results of testing at one location were more consistent, generally producing higher "R" values than results obtained by varying the test location.

A.3 - SUMMARY OF CONCRETE STRENGTH TESTING RESULTS

In general, the results of the various strength tests on the concrete test specimens indicated that the mix designs in Table A-2 met the specifications of Table 4-2, and were acceptable for using drop test experiments.

Table A-3 summarizes the results of UCS tests for the trial batch and test specimen batch test cylinders; Table A-4 summarizes the results of ITS tests. In these tables, the "LS" series tests represent low strength concrete, while "HS" represents high strength. In general, UCS for the test specimen batch was higher compared to the trial batch. This result can be attributed to the fact that fresh Type III cement was used in the test specimen batch. The cement used in the trial batch was several years old and therefore, less reactive.

It is worth noting that the low strength test cylinders experienced a greater UCS increase between the trial and test specimen batches than the high strength cylinders. As a result, the trial batch had a larger compressive strength ratio (i.e., the ratio of the high strength to low strength mix UCS was 3.6 at 28 days) than the test specimen batch (2.7 at 28 days). This result can be attributed to a slight variation in moisture content for the low strength mix between the trial and test specimen batches. The low strength mix for the trial batch was also observed to be wetter (higher slump value) than the test specimen batch. Since less water was apparently present in the low strength mix of the test specimen batch, a higher strength resulted. Even though the compressive strength ratio was lower for the test specimen batch, it was still well within the limits specified in Table 4-1 (i.e., in the range of 2 to 3).

Tensile strength tests were performed on a batch of concrete prepared specifically for those tests. The ITS for test cylinders ranged from about 5% of the UCS for the high strength mix, to about 10% of the UCS for the low strength mix. The tensile strength ratio (Table A-4) between the low and high strength mixes was somewhat less (1.6 at 28 days) than the compressive strength ratio. This indicates that the high and low strength specimens probably had different brittleness characteristics.

TABLE A-3 - Concrete Testing Results: Unconfined Compression Tests

COMPRESSIVE STRENGTH TESTS											
Sample No. (1)	Mix Date	14 Day Break Date	UCS (psi)	Average 14 Day UCS (psi)	Strength Ratio (5)	Sample No. (1)	Mix Date	28 Day Break Date	UCS (psi)	Average 28 Day UCS (psi)	Strength Ratio (5)
TRIAL BATCH											
LS1	5/30/95	6/13/95 (2)	2225	2313	3.9	LS3	5/30/95	6/26/95	2672	2628	3.6
LS2	5/30/95	6/13/95 (2)	2400			LS4	5/30/95	6/26/95	2584		
HS1	6/6/95	6/22/95 (3)	7148 (4)	9011		HS3	6/6/95	7/3/95	5264 (4)	9460	
HS2	6/6/95	6/22/95 (3)	9011			HS4	6/6/95	7/3/95	9460		
TEST SPECIMEN BATCH											
LS5	6/15/95	6/29/95	2919	3489	2.7	LS7	6/29/95	7/13/95	3239	3624	2.7
LS6	6/15/95	6/29/95	3489			LS8	6/29/95	7/13/95	4008		
HS5	6/16/95	6/30/95	9423	9423		HS7	6/30/95	7/14/95	10010	9915	
HS6	6/16/95	6/30/95	8070 (4)			HS8	6/30/95	7/14/95	9820		

(1) HS = High strength, LS = Low strength

(2) This is a 15 day test due to scheduling constraints

(3) This is a 16 day test due to scheduling constraints

(4) This strength value is not representative due to voids left in sample during placement

(5) Strength ratio is the ratio of the high to low strength concrete

All tests performed on 4-inch diameter by 8-inch long cylinders in accordance with ASTM C39

TABLE A-4 - Concrete Testing Results: Tensile Tests

TENSILE TESTS

TENSILE TESTS											
Sample No.	Mix	14 Day Break	ITS (2)	Average 14 Day	Strength	Sample No.	Mix	28 Day Break	ITS (2)	Average 28 Day	Strength
(1)	Date	Date	(psi)	ITS (psi)	Ratio (3)	(1)	Date	Date	(psi)	ITS (psi)	Ratio (3)
LS9	9/27/95	10/11/95	315	291	1.9	LS11	9/27/95	10/25/95	354	360	1.6
LS10	9/27/95	10/11/95	267			LS12	9/27/95	10/25/95	366		
HS9	9/27/95	10/11/95	587	561		HS11	9/27/95	10/25/95	559	572	
HS10	9/27/95	10/11/95	534			HS12	9/27/95	10/25/95	585		

(1) HS = High strength; LS = Low strength

(2) ITS = Indirect tensile strength

(3) Strength ratio is the ratio of the high to low strength concrete

All tests performed on 4-inch diameter by 8-inch long cylinders in accordance with ASTM C496

In general, the low strength concrete cylinders failed in a non-catastrophic, semi-brittle manner in the UCS tests. Bulging was noted in the center of most of the low strength specimens during UCS the tests, resulting in a "damaged zone" rather than a distinct failure surface. In the ITS tests, a distinct failure surface was formed along the axis of the test cylinders, but failure generally occurred in a non-catastrophic manner, without an accompanying sharp noise. In contrast, the high strength concrete cylinders generally failed in a more brittle manner in the UCS tests, and was accompanied by a sharp noise and the ejection of fragments at failure. Well-defined, high angle (75-90°) failure surfaces were also noted in most of these specimens.

Voids were observed in some of the high strength specimens, obviously left from rodding the wet concrete when casting the test cylinders. It was clear that these voids caused premature failure of some test cylinders in the compression tests, as noted in Table A-3. Failure of the high strength specimens in the Brazilian tensile tests was semi-brittle. The failure surface developed parallel to the axis of loading, and was generally accompanied by a sharp noise.

Figures A-3 and A-4 present the results of non-destructive concrete tests, including plots of dynamic Young's modulus and UCS versus time.

**FIGURE A-3 - Non-Destructive Testing Results:
Dynamic Young's Modulus of Concrete Tests Specimens**

Inspection of these plots reveals that the concrete cured at a rapid rate, with a majority of the strength being achieved within two weeks time. This was expected, since high early strength Type III cement was used. Figure A-3 suggests that the ratio of the dynamic Young's modulus between high and low strength specimens is just under two. The lower strength specimens are approximately 1/2 to 1/3 the UCS of the high strength specimens, as was expected from the development of the mix designs (Figure A-4). It is also noteworthy that the values of UCS determined by non-destructive tests are very close to those determined from the direct UCS tests (Table A-3). The values of dynamic Young's modulus, however, are suspect since they are higher than many competent rocks. Figure A-4 illustrates a comparison of estimated UCS for the low and high strength specimens.

Figure A-5 presents a plot of Schmidt hammer test results versus time for low strength and high strength concrete specimens. On this plot, the effect on the average Schmidt hardness of testing at varied locations on the specimens vs. a singular location is illustrated. The increase in specimen Schmidt hardness generally increases at the same rate as the UCS in Figure A-3.

**FIGURE A-4 - Non-Destructive Testing Results:
UCS of Concrete Tests Specimens**

FIGURE A-5 - Schmidt Hammer Test Results on Concrete Test Specimens

APPENDIX B

IMPACT DATA: REPRESENTATIVE TIME HISTORIES

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Table B-1 summarizes the channel assignments for data aquisitioning, and scaling factors for converting the data from voltage to engineering units.

TABLE B-1 - Data Aquisitioning System Channel Assignments

CHANNEL	FUNCTION	SCALE FACTOR
1	Force: 2:00 Position	7751.94 lb/volt
2	Force: 10:00 Position	7751.94 lb/volt
3	Force: 4:00 Position	7575.76 lb/volt
4	Force: 8:00 Position	7575.76 lb/volt
5	Acceleration: CUTTER	100 g's per volt
6	Acceleration: ROCK FIXTURE	10 g's per volt
7	Displacement: MAIN CARRIAGE	3.785534 in. / volt
8	Displacement: CUTTER	Polynomial (1)

(1) Scale Factor based on polynomial fit:

$$\text{in} = 11.11928v - 66.5342v^{1/2} + 78.15401v^{1/3}$$

where v = volts

LIST OF ABBREVIATIONS

The abbreviations found in the charts in this appendix are defined as follows:

- "cut" forces = cutter side forces in the direction parallel to impact rows
- "left to right" (L to R) forces = cutter side forces that are perpendicular to the impact rows.
- "Cutter" acceleration = vibrational acceleration experienced by the cutter.
- "Box" acceleration = vibrational acceleration experienced by the specimen containment box.
- "First impact" = the main impact of the cutter on a specimen
- "Penetration time" = the time from initial cutter contact to the time of maximum normal cutter force.
- "Cutter rebound time" = time from first cutter impact to the rebound impact.

Normal Force vs. Time

Side Forces vs. Time

Cutter Rebound Time:
= 0.16 sec

Average
Residual= 0.11079 in

Maximum
Residual= 0.31340 in

Minimum
Residual= -0.24327 in

Cutter Rebound Time:
= 0.20 sec

Average
Residual = -0.02440 in

Maximum
Residual = 0.16108 in

Minimum
Residual = -0.37525 in

Acceleration vs. Time

Cutter Displacement vs. Time

Area Under Curve = Impulse = 173 lb-sec
 Maximum Normal Force = 40127 lb
 Average Normal Force = 1155 lb

Penetration Time = 0.0024 sec

Maximum Side Force (cut) = 3766 lb
 Maximum Side Force (L to R) = 2961 lb

Maximum Cutter Acceleration = 392 g's
 Maximum Box Acceleration = 1 g's

Note: Box Accelerometer Not Connected

Cutter Rebound Time:
= 0.25 sec.

Acceleration vs. Time

Cutter Displacement vs. Time

Cutter Rebound Time:
= 0.15 sec

Cutter Rebound Time:
=0.19 sec

Area Under Curve = Impulse = 445 lb-sec.
Maximum Normal Force = 39049 lb.

Penetration Time = 0.0034 sec

Maximum Side Force (cut) = 3534 lb
Maximum Side Force (L to R) = 1104 lb

Maximum Cutter Acceleration = 324 g's
Maximum Box Acceleration = 63 g's

APPENDIX C

SUMMARIZED DATA:

VELOCITY

IMPACT CHARACTERISTICS

PENETRATION

SPECIFIC ENERGY

TABLE C-1 - Velocity Calculations for Individual Impact Tests

CONCRETE SPECIMENS						
	Test Number	Row Spacing, in	Impact Set	Impact Position	VELOCITY AT IMPACT, ft/sec (1)	
					Slope	Differentiation (2)
Low Velocity	RDLW1A1	0	1	1	9.5	9.5
	RDLW1A2	0	1	2	9.4	9.3
	RDLW1E2	3	1	2	9.3	9.5
	RDLW1E22	3	2	2	9.2	9.5
	RDLW1D2	3	1	2	9.2	9.2
	RDLW1D22	3	2	2	9.1	9.6
	RDLS1A2	0	1	2	9.5	9.4
	RDLS1E2	3	1	2	9.5	9.6
	RDLS1E22	3	2	2	9.3	9.4
	RDLS1E32	3	3	2	9.4	9.7
	RDLS1E42	3	4	2	9.2	9.5
	RDLS1D3	3	1	3	9.7	9.5
	RDLS1D22	3	2	3	9.3	9.1
	Average:				9.3	9.4
	Maximum:				9.7	9.7
	Standard Deviation:				0.2	0.2
High Velocity	RDHW2E2	3	1	2	18.9	NM
	RDHW2E22	3	2	2	18.4	19.0
	RDHW2D2	3	1	2	19.1	19.1
	RDHW2D22	3	2	2	17.6	18.9
	RDHW2E3	3	2	3	18.7	NM
	RDHW2E23	3	2	3	18.3	NM
	RDHS2A2	0	1	2	18.9	18.9
	RDHS2E2	3	1	2	19.2	18.4
	RDHS2E22	3	2	2	18.8	19.0
	RDHS2D2	3	1	2	19.2	19.0
	RDHS2D22	3	2	2	19.5	19.0
	RDHS2E3	3	2	3	19.3	NM
	RDHS2E23	3	2	3	19.1	NM
	Average:				18.9	18.9
	Maximum:				19.5	19.1
	Standard Deviation:				0.5	0.2

(1) Newton method curve fit was performed resulting in:

Low velocity = 9.9 ft/sec

High velocity = 19.0 ft/sec

(2) Numerical differentiation of displacement - time history

(3) NM = Not Measured

TABLE C-1 (continued) - Velocity Calculations for Individual Impact Tests 130

ROCK SPECIMENS						
Low Velocity	Test Number	Row Spacing, in	Impact Set	Impact Position	VELOCITY AT IMPACT, ft/sec (1)	
					Slope	Differentiation (2)
	RDLBGA2	0	1	2	10.16	9.5
	RDLBGA22	0	2	2	9.98	9.7
	RDLBGA32	0	3	2	10.60	9.7
	RDLBGB2	2	1	2	9.80	9.7
	RDLBGB22	2	2	2	9.16	9.5
	RDLBGB32	2	3	2	9.26	9.8
	Average:				9.8	9.7
	Maximum:				10.6	9.8
	Standard Deviation:				0.5	0.1

(1) Newton method curve fit was performed resulting in:

Low velocity = 9.9 ft/sec

(2) Numerical differentiation of displacement - time history

(3) NM = Not Measured

TABLE C-2 - Impact Characteristics for Individual Impact Tests

CONCRETE SPECIMENS																
		Test Number	Row Spacing, in	Impact Set	Position	Maximum Impact Force, lb			Impulse lb-sec	Maximum Acceleration (cutter), g's	Maximum Acceleration (box), g's	Damping Time, sec.	Rebound Time, sec	Coef of Restitution	Energy Transfer Coef.	
						Normal	(L to R)	(cut)								
LOW VELOCITY V = 9.3 ft/sec Cutter Weight - 485 lb	Low Strength UCS = 3624 psi	RDLW1A1	0	1	1	40,673	1,714	3,294	45	117	51	0.008	0.183	0.29	0.92	
		RDLW1A2	0	1	2	33,555	905	1,962	46	117	98	0.009	0.166	0.26	0.93	
		RDLW1A3	0	1	3	31,774	2,798	1,968	43	118	67	0.010	0.172	0.27	0.93	
		RDLW1B1	2	1	1	38,620	1,125	1,674	48	150	41	0.010	0.160	0.25	0.94	
		RDLW1B2	2	1	2	28,228	2,294	1,148	43	123	71	0.010	0.160	0.25	0.94	
		RDLW1C1	4	1	1	41,540	2,924	1,224	47	154	26	0.010	0.197	0.31	0.90	
		RDLW1C2	4	1	2	38,492	1,790	1,214	47	278	30	0.010	0.206	0.32	0.90	
		RDLW1C3	4	1	3	37,916	1,308	1,037	48	262	45	0.010	0.208	0.33	0.89	
		RDLW1D2	3	1	2	35,298	2,151	7,404	39	195	67	0.009	0.203	0.32	0.90	
		RDLW1D22	3	2	2	35,475	1,415	5,945	40	157	59	0.011	0.196	0.31	0.91	
		RDLW1E2	3	1	2	29,495	1,109	2,871	44	250	34	0.010	0.214	0.34	0.89	
		RDLW1E22	3	2	2	37,642	4,298	7,515	37	167	59	0.009	0.186	0.29	0.91	
		Average:					35,726	1,986	3,105	44	174	54	0.010	0.188	0.29	0.91
		Maximum:					41,540	4,298	7,515	48	278	98	0.011	0.214	0.34	0.94
		Standard Deviation:					4,247	981	2,449	4	59	21	0.001	0.019	0.03	0.02
Impact Energy: = 651 ft-lb (per button) = 217 ft-lb	High Strength UCS = 9,916 psi	RDL51A2	0	1	2	44,590	2,840	2,937	46	334	(1)	0.009	0.195	0.31	0.91	
		RDL51A3	0	3	1	42,631	3,247	1,432	43	290	67	0.010	0.197	0.31	0.90	
		RDL51B1	2	1	1	46,787	3,388	3,667	46	487	98	0.009	0.178	0.28	0.92	
		RDL51B2	2	2	1	47,917	3,781	1,770	46	677	67	0.009	0.104	0.16	0.97	
		RDL51C21	4	1	1	58,841	3,749	1,677	41	678	51	0.009	0.174	0.27	0.93	
		RDL51C22	4	2	2	33,375	4,064	3,366	35	403	98	0.010	0.153	0.24	0.94	
		RDL51D3	3	1	3	36,718	5,012	3,108	46	538	40	0.009	0.195	0.31	0.91	
		RDL51D22	3	2	3	34,500	2,774	3,890	43	360	48	0.011	0.164	0.26	0.93	
		RDL51E2	3	1	2	46,196	1,818	4,446	48	847	44	0.010	0.204	0.32	0.90	
		RDL51E22	3	2	2	38,495	3,185	5,194	41	271	83	0.010	0.180	0.28	0.92	
		RDL51E32	3	3	2	58,876	4,384	5,133	39	660	84	0.009	0.171	0.27	0.93	
		RDL51E42	3	4	2	53,231	3,412	2,802	50	374	64	0.009	0.217	0.34	0.88	
		Average:					45,180	3,471	3,285	44	495	68	0.010	0.178	0.28	0.92
		Maximum:					58,876	5,012	5,194	50	847	98	0.011	0.217	0.34	0.97
		Standard Deviation:					8,669	825	1,266	4	184	21	0.001	0.029	0.05	0.03

(1) Accelerometer not functioning or out of range

(2) No response after impact: no rebound measurement

TABLE C-2 - Impact Characteristics for Individual Impact Tests

CONCRETE SPECIMENS																
	Test Number	Row Spacing, in	Impact Set	Position	Maximum Impact Force, lb			Impulse lb-sec	Maximum Acceleration (cutter), g's	Maximum Acceleration (box), g's	Damping Time, sec	Rebound Time, sec	Coef of Restitution	Energy Transfer Coef.		
					Normal	(L to R)	(cut)									
HIGH VELOCITY V = 18.9 ft/sec Cutter Weight = 130 lb	Low Strength UCS = 3624 psi	RDHW2B21	2	2	1	47,897	1,276	3,522	17	551	(1)	0.004	0.216	0.17	0.97	
		RDHW2B22	2	2	2	55,395	1,811	2,578	16	379	(1)	0.004	0.156	0.12	0.99	
		RDHW2B23	2	2	3	27,850	719	2,829	16	389	(1)	0.004	0.154	0.12	0.99	
		RDHW2C1	4	1	1	38,636	483	561	17	548	(1)	0.003	0.226	0.18	0.97	
		RDHW2C2	4	1	2	31,197	651	884	16	512	(1)	0.006	0.197	0.15	0.98	
		RDHW2D1	3	1	1	35,007	797	818	17	452	(1)	0.003	0.225	0.17	0.97	
		RDHW2D2	3	1	2	30,851	2,037	735	17	310	(1)	0.005	0.206	0.16	0.97	
		RDHW2D22	3	2	2	51,084	1,682	2,896	19	681	(1)	0.006	0.254	0.20	0.96	
		RDHW2E2	3	1	2	31,726	1,143	1,454	15	400	(1)	0.004	0.214	0.17	0.97	
		RDHW2E3	3	2	3	25,574	2,082	1,004	16	359	(1)	0.005	0.184	0.14	0.98	
	RDHW2E22	3	2	2	40,127	2,961	3,766	17	392	(1)	0.005	0.250	0.19	0.96		
	RDHW2E23	3	2	3	36,437	3,426	515	17	459	(1)	0.005	0.244	0.19	0.96		
	Average:					35,982	1,589	1,797	17	453	---	0.005	0.211	0.16	0.97	
	Maximum:					51,084	3,426	3,766	19	681	---	0.006	0.254	0.20	0.99	
	Standard Deviation:					7,613	931	1,227	1	184	---	0.001	0.033	0.03	0.01	
	Impact Energy = 721 ft-lb (per button) = 240 ft-lb	High Strength UCS = 9,916 psi	RDHS2A2	0	1	2	27,455	1,762	1,519	16	912	(1)	0.006	0.152	0.12	0.99
			RDHS2A3	0	1	3	24,841	1,759	1,774	15	336	(1)	0.007	0.185	0.14	0.98
			RDHS2B1	2	1	1	31,769	1,368	2,386	15	397	(1)	0.004	0.113	0.09	0.99
			RDHS2B22	2	1	2	39,764	1,090	3,573	16	531	90	0.004	0.102	0.08	0.99
			RDHS2B21	2	2	1	56,765	1,438	5,668	17	1000	98	0.003	0.272	0.21	0.96
RDHS2C1			4	1	1	44,172	1,716	2,094	17	541	98	0.003	0.234	0.18	0.97	
RDHS2C2			4	1	2	36,430	3,017	1,305	18	699	96	0.004	0.184	0.14	0.98	
RDHS2D2			3	1	2	32,364	1,504	3,377	16	626	98	0.008	0.199	0.15	0.98	
RDHS2D22			3	2	2	49,467	7,104	1,766	16	481	98	0.007	0.245	0.19	0.96	
RDHS2E2			3	1	2	36,361	4,816	4,901	16	610	98	0.008	(2)	--	--	
RDHS2E3			3	2	3	27,792	1,447	1,512	16	373	98	0.006	0.154	0.12	0.99	
RDHS2E22			3	2	2	35,699	3,636	2,789	17	429	98	0.007	0.228	0.18	0.97	
RDHS2E23			3	2	3	56,414	1,764	5,381	17	771	98	0.006	0.215	0.17	0.97	
Average:					38,409	2,494	2,927	16	608	97	0.006	0.190	0.15	0.98		
Maximum:					56,765	7,104	5,668	18	1,000	98	0.008	0.272	0.21	0.99		
Standard Deviation:					10,533	1,753	1,537	1	198	3	0.002	0.052	0.04	0.01		

- (1) Bare accelerometer not functioning or out of range
(2) No response after impact: no rebound measurement

TABLE C-2 (continued) - Impact Characteristics for Individual Impact Tests

ROCK SPECIMENS														
	Test Number	Row Spacing, in	Impact Set	Position	Maximum Impact Force, lb			Impulse lb-sec	Maximum Acceleration (cutter), g's	Maximum Acceleration (box), g's	Damping Time, sec	Rebound Time, sec	Coef of Restitution	Energy Transfer Coef
					Normal	(L to R)	(cut)							
LOW VELOCITY V = 9.8 f/sec Cutter Weight = 485 lb Impact Energy = 723 ft-lb (per button) = 241 ft-lb	Berea Sandstone UCS = 6,700 psi	RPLBSA1	0	1	54,159	5,378	1,546	49	467	85	0.008	0.216	0.34	0.88
		RPLBSA2	0	1	39,049	1,104	3,534	45	324	50	0.012	0.138	0.22	0.95
		RPLBSA3	0	1	40,167	1,859	3,386	44	500	47	0.011	0.120	0.19	0.96
		RPLBSA21	3	2	75,290	3,219	5,625	50	1,000	73	0.008	0.226	0.36	0.87
		RPLBSA22	3	2	61,680	3,660	5,771	50	850	75	0.009	0.211	0.33	0.89
		RPLBSA23	3	2	45,077	2,648	3,126	49	636	82	0.008	0.198	0.31	0.90
		RPLBSB1	3	1	53,999	1,329	5,182	50	996	57	0.007	0.223	0.35	0.88
		RPLBSB2	3	1	39,790	1,306	3,396	44	674	115	0.010	0.121	0.19	0.96
		RPLBSB3	3	1	35,649	1,386	2,826	45	516	43	0.010	0.140	0.22	0.95
		RPLBSB21	3	2	65,926	3,699	3,778	50	1,000	65	0.007	0.221	0.35	0.88
		RPLBSB22	3	1	55,136	635	1,766	50	816	75	0.008	0.208	0.33	0.89
		RPLBSB23	3	2	50,768	2,122	2,011	50	715	75	0.008	0.213	0.33	0.89
		Average:			51,391	2,362	3,496	48	708	70	0.009	0.186	0.29	0.91
		Maximum:			75,290	5,378	5,771	50	1,000	115	0.012	0.226	0.36	0.96
		Standard Deviation:			12,127	1,395	1,422	3	229	20	0.002	0.043	0.07	0.04
	Barre Granite UCS = 28,600 psi	RDLBGA2	0	1	54,163	2,315	3,375	47	1,000	(1)	0.013	0.156	0.25	0.94
		RDLBGA22	0	2	76,022	3,379	3,162	50	831	98	0.011	0.193	0.30	0.91
		RDLBGA32	0	3	72,039	3,926	5,683	49	1,000	98	0.010	0.205	0.32	0.90
		RDLBGB2	2	1	44,840	3,702	5,249	46	1,000	59	0.010	0.188	0.30	0.91
		RDLBGB22	2	2	75,407	2,192	3,203	50	770	92	0.008	0.207	0.33	0.89
		RDLBGB32	2	3	84,676	3,056	4,722	48	870	98	0.008	0.185	0.29	0.92
		RPLBGA1	0	1	85,207	5,102	4,170	47	(1)	200	0.008	0.153	0.24	0.94
		RPLBGA2	0	1	64,506	5,538	4,300	53	(1)	200	0.010	0.154	0.24	0.94
		RPLBGB1	1.5	1	61,296	6,180	4,925	54	(1)	132	0.009	0.191	0.30	0.91
		RPLBGB2	1.5	1	63,027	5,281	4,592	56	(1)	194	0.011	0.167	0.26	0.93
		RPLBGC1	1.5	1	90,723	3,389	4,291	59	(1)	250	0.008	0.179	0.28	0.92
		RPLBGC2	1.5	1	66,990	5,612	3,280	57	(1)	195	0.012	0.140	0.22	0.95
		Average:			69,908	4,139	4,246	51	912	147	0.010	0.177	0.28	0.92
		Maximum:			90,723	6,180	5,683	59	1,000	250	0.013	0.207	0.33	0.95
		Standard Deviation:			13,461	1,355	844	4	102	62	0.002	0.022	0.03	0.02

(1) Accelerometer not functioning or out of range

(2) No response after impact: no rebound measurement

TABLE C-3 - Penetration Predicted by Analytical Models

a) Fuchs Method:

Impact Velocity	Specimen	Mass of cutter per button, slug	Impact velocity, ft/sec	Schmidt Hardness, R	Specimen Density, pcf	Average Maximum Impact Force, lb	α	β	Predicted Penetration, ft	Predicted Penetration, in
Low	LS	5.02	9.3	39	135	35356	11965	0.004288	0.018	0.22
	HS	5.02	9.3	52	142	44658	15953	0.004510	0.014	0.16
High	LS	1.35	18.9	39	135	35967	11965	0.004288	0.020	0.24
	HS	1.35	18.9	52	142	37936	15953	0.004510	0.015	0.18
Low	Berea SS	5.02	9.8	52	132	40000	15953	0.004192	0.015	0.18
Low	Barre Gr	5.02	9.8	70	164	67858	21476	0.005209	0.011	0.13

b) Young's Method:

Velocity	Strength	Mass of cutter per button, kg	K	Schmidt Hardness, R	S	Impact velocity, ft/sec	Impact velocity, m/s	Predicted Penetration, cm	Predicted Penetration, in
Low	LS	73.50	1.42	39	1.22	9.3	2.8	0.499	0.20
	HS	73.50	1.42	52	0.96	9.3	2.8	0.392	0.15
High	LS	20.00	1.20	39	1.22	18.9	5.8	0.447	0.18
	HS	20.00	1.20	52	0.96	18.9	5.8	0.351	0.14
Low	Berea SS	73.50	1.42	52	0.96	9.8	3.0	0.413	0.16
Low	Barre Gr.	73.50	1.42	70	0.60	9.8	3.0	0.258	0.10

LS = Low strength concrete specimen

HS = High strength concrete specimen

SS = Sandstone

S = (100-R)*C, where C = 0.02

TABLE C-4 - Penetration Calculations for Individual Impact Tests

LOW VELOCITY TESTS ON CONCRETE SPECIMENS

	Test Number	Time of Penetration, sec	Calculated Penetration, in		Test Number	Time of Penetration, sec	Calculated Penetration, in
Low Strength UCS = 3624 psi	RDLW1A1	0.0040	0.22	High Strength UCS = 9916 psi	RDLS1A1	0.0024	0.13
	RDLW1A2	0.0032	0.18		RDLS1A2	0.0020	0.11
	RDLW1A21	0.0030	0.17		RDLS1A21	0.0012	0.07
	RDLW1A23	0.0044	0.25		RDLS1A23	0.0040	0.22
	RDLW1B1	0.0040	0.22		RDLS1B1	0.0020	0.11
	RDLW1B2	0.0044	0.25		RDLS1C21	0.0018	0.10
	RDLW1C1	0.0034	0.19		RDLS1C22	0.0036	0.20
	RDLW1C3	0.0050	0.28		RDLS1C23	0.0044	0.25
	RDLW1C21	0.0024	0.13		RDLS1D3	0.0026	0.15
	RDLW1C22	0.0036	0.20		RDLS1D21	0.0016	0.09
	RDLW1C23	0.0052	0.29		RDLS1D22	0.0032	0.18
	RDLW1D2	0.0040	0.22		RDLS1D23	0.0032	0.18
	RDLW1D21	0.0030	0.17		RDLS1E1	0.0024	0.13
	RDLW1D22	0.0044	0.25		RDLS1E2	0.0024	0.13
	RDLW1E2	0.0040	0.22		RDLS1E3	0.0016	0.09
	RDLW1E3	0.0042	0.23		RDLS1E21	0.0020	0.11
	RDLW1E21	0.0036	0.20		RDLS1E22	0.0036	0.20
	RDLW1E22	0.0032	0.18		RDLS1E32	0.0020	0.11
	RDLW1E23	0.0048	0.27		RDLS1E42	0.0032	0.18
	Average:	0.0039	0.22		Average:	0.0026	0.14
	Maximum:	0.0052	0.29		Maximum:	0.0044	0.25
	Standard Deviation:	0.0007	0.04		Standard Deviation:	0.0009	0.05

TABLE C-4 (continued) - Penetration Calculations for Individual Impact Tests

HIGH VELOCITY TESTS ON CONCRETE SPECIMENS							
	Test Number	Time of Penetration, sec	Calculated Penetration, in		Test Number	Time of Penetration, sec	Calculated Penetration, in
Low Strength UCS = 3624 psi	RDHW2B21	0.0012	0.14	High Strength UCS = 9916 psi	RDHS2A1	0.0016	0.18
	RDHW2B22	0.0024	0.27		RDHS2A2	0.0028	0.32
	RDHW2B23	0.0028	0.32		RDHS2A3	0.0024	0.27
	RDHW2C1	0.0020	0.23		RDHS2A21	0.0014	0.16
	RDHW2C2	0.0020	0.23		RDHS2A23	0.0016	0.18
	RDHW2C21	0.0016	0.18		RDHS2B1	0.0014	0.16
	RDHW2C22	0.0020	0.23		RDHS2B3	0.0014	0.16
	RDHW2C23	0.0020	0.23		RDHS2B21	0.0012	0.14
	RDHW2D1	0.0016	0.18		RDHS2B22	0.0012	0.14
	RDHW2D2	0.0016	0.18		RDHS2C1	0.0018	0.20
	RDHW2D3	0.0020	0.23		RDHS2C2	0.0016	0.18
	RDHW2D21	0.0012	0.14		RDHS2C3	0.0012	0.14
	RDHW2D22	0.0016	0.18		RDHS2C21	0.0016	0.18
	RDHW2D23	0.0016	0.18		RDHS2C22	0.0018	0.20
	RDHW2E1	0.0020	0.23		RDHS2C23	0.0020	0.23
	RDHW2E2	0.0020	0.23		RDHS2D1	0.0016	0.18
	RDHW2E3	0.0030	0.34		RDHS2D2	0.0012	0.14
	RDHW2E21	0.0016	0.18		RDHS2D3	0.0016	0.18
	RDHW2E22	0.0024	0.27		RDHS2D21	0.0020	0.23
	RDHW2E23	0.0024	0.27		RDHS2D22	0.0024	0.27
				RDHS2E1	0.0016	0.18	
				RDHS2E2	0.0012	0.14	
				RDHS2E3	0.0018	0.20	
				RDHS2E21	0.0014	0.16	
				RDHS2E22	0.0030	0.34	
				RDHS2E23	0.0020	0.23	
Average:		0.0020	0.22	Average:		0.0017	0.20
Maximum:		0.0030	0.34	Maximum:		0.0030	0.34
Standard Deviation:		0.0005	0.05	Standard Deviation:		0.0005	0.05

TABLE C-4 (continued) - Penetration Calculations for Individual Impact Tests

LOW VELOCITY TEST ON ROCK SPECIMENS

	Test Number	Time of Penetration, sec	Calculated Penetration, in		Test Number	Time of Penetration, sec	Calculated Penetration, in
Barre Granite UCS = 28,600 psi	RDLBGA1	0.0020	0.12	Berea Sandstone UCS = 6,700 psi	RPLBSA1	0.0021	0.12
	RDLBGA2	0.0020	0.12		RPLBSA2	0.0034	0.20
	RDLBGA3	0.0020	0.12		RPLBSA3	0.0020	0.12
	RDLBGA21	0.0012	0.07		RPLBSA21	0.0022	0.13
	RDLBGA22	0.0012	0.07		RPLBSA22	0.0024	0.14
	RDLBGA31	0.0016	0.09		RPLBSA23	0.0024	0.14
	RDLBGA32	0.0020	0.12		RPLBSB1	0.0021	0.12
	RDLBGA33	0.0016	0.09		RPLBSB2	0.0024	0.14
	RDLBGB1	0.0020	0.12		RPLBSB3	0.0046	0.27
	RDLBGB2	0.0020	0.12		RPLBSB21	0.0017	0.10
	RDLBGB3	0.0016	0.09		RPLBSB22	0.0014	0.08
	RDLBGB21	0.0016	0.09		RPLBSB23	0.0027	0.16
	RDLBGB22	0.0020	0.12				
	RDLBGB31	0.0012	0.07				
	RDLBGB32	0.0016	0.09				
	RDLBGB33	0.0016	0.09				
	Average:	0.0017	0.10		Average:	0.0025	0.14
	Maximum:	0.002	0.12		Maximum:	0.005	0.27
	Standard Deviation:	0.000	0.02		Standard Deviation:	0.001	0.05

TABLE C-5 - Specific Energy Calculations

CONCRETE SPECIMENS - ROW DIMENSIONING TESTS

		Test Series	Row Spacing, in	Impact Set	Cutting Gradation (1)	Weight of Cuttings, g	Weight of Cuttings, ton	Specific Energy Hp-hr/ton	Volume (2) of Crater, cc	Volume (2) of Crater, cu yd	Specific Energy Hp-hr/cu yd	Volume (3) of Cuttings, cc	Volume (3) of Cuttings, cu yd	Specific Energy Hp-hr/cu yd
LOW VELOCITY V = 9.3 ft/sec Cutter Weight = 485 lb Impact Energy = 651 ft-lb (per button) = 217 ft-lb = 1.10E-04 Hp-hr	Low Strength UCS = 3624 psi	RDLW1A1-3	0	1	WG	26.1	2.9E-05	11.4	--	--	--	24	3.1E-05	10.5
		RDLW1B1-3	2	1	GG	164.4	1.8E-04	1.8	150	2.0E-04	1.7	68	8.9E-05	3.7
		RDLW1C1-3	4	1	--	17.5	1.9E-05	17.1	--	--	--	--	--	--
		RDLW1C21-3	4	2	--	83.2	9.2E-05	5.9	--	--	--	--	--	--
		RDLW1D1-3A21-3	3.5	1	--	56.1	6.2E-05	10.6	--	--	--	--	--	--
		RDLW1E1-3	3	1	--	18.9	2.1E-05	15.8	--	--	--	9	1.2E-05	27.9
	High Strength UCS = 9,916 psi	RDLW1E21-3	3	2	GG	804.5	8.9E-04	0.7	550	7.2E-04	0.9	413	5.4E-04	1.2
		RDL51A1-3	0	1	WG	11.9	1.3E-05	25.1	--	--	--	5	5.9E-06	55.9
		RDL51A21-3	0	2	WG	31.6	3.5E-05	13.7	--	--	--	14	1.9E-05	26.7
		RDL51B1-3	2	1	--	167.1	1.8E-04	1.8	--	--	--	72	9.4E-05	3.5
		RDL51B21-3	2	2	GG	76.7	8.5E-05	2.4	135	1.8E-04	3.7	32	4.1E-05	4.9
		RDL51C1-3	4	1	--	48.3	5.3E-05	6.2	--	--	--	--	--	--
		RDL51C21-3	4	2	--	31.2	3.4E-05	7.5	--	--	--	--	--	--
		RDL51D1-3	3.5	1	--	26.5	2.9E-05	11.3	--	--	--	--	--	--
		RDL51D21-3	3.5	2	--	54.9	6.1E-05	7.3	--	--	--	--	--	--
		RDL51D31-3	3.5	3	--	21.0	2.3E-05	8.7	--	--	--	--	--	--
		RDL51D41-3	3.5	4	--	33.2	3.7E-05	8.8	--	--	--	--	--	--
		RDL51E1-3	3	1	--	27.3	3.0E-05	10.9	--	--	--	12	1.5E-05	21.9
		RDL51E21-3	3	2	--	55.6	6.1E-05	7.2	--	--	--	24	3.1E-05	14.2
		RDL51E31-3	3	3	--	29.1	3.2E-05	8.0	--	--	--	12	1.6E-05	15.9
		RDL51E41-3	3	4	WG	29.5	3.3E-05	8.4	--	--	--	14	1.8E-05	16.5
HIGH VELOCITY V = 18.9 ft/sec Cutter Weight = 130 lb Impact Energy = 721 ft-lb (per button) = 240 ft-lb = 1.21E-04 Hp-hr	Low Strength UCS = 3624 psi	RDHW2C1-3	4	1	--	50.4	5.6E-05	6.5	--	--	--	--	--	--
		RDHW2C21-3	4	2	--	71.9	7.9E-05	5.4	--	--	--	--	--	--
		RDHW2D1-3	3.5	1	--	34.3	3.8E-05	9.6	--	--	--	--	--	--
		RDHW2D21-3	3.5	2	--	49.9	5.5E-05	7.8	--	--	--	--	--	--
		RDHW2E1-3	3	1	--	220.2	2.4E-04	1.5	--	--	--	97	1.3E-04	2.6
		RDHW2E21-3	3	2	WG	183.4	2.0E-04	1.6	260	3.4E-04	1.9	85	1.1E-04	2.8
	High Strength UCS = 9,916 psi	RDHS2B1-3	2	1	--	151.3	1.7E-04	2.2	--	--	--	63	8.2E-05	4.0
		RDHS2B21-3	2	2	WG	197.4	2.2E-04	1.9	--	--	--	79	1.0E-04	3.5
		RDHS2C1-3	4	1	--	77.0	8.5E-05	4.3	--	--	--	--	--	--
		RDHS2C21-3	4	2	--	52.0	5.7E-05	5.1	--	--	--	--	--	--
		RDHS2D1-3	3.5	1	--	43.0	4.7E-05	7.7	--	--	--	19	2.5E-05	13.5
		RDHS2D21-3	3.5	2	GG	382.1	4.2E-04	1.6	470	6.1E-04	1.1	166	2.2E-04	4.1
		RDHS2E1-3	3	1	--	69.0	7.6E-05	4.8	--	--	--	28	3.7E-05	9.0
		RDHS2E21-3	3	2	GG	384.3	4.2E-04	1.5	455	6.0E-04	1.1	165	2.2E-04	2.6

(1) WG = Well Graded, GG = Gap Graded, PG = Poorly Graded

(2) Volume measured using glycerin

(3) Volume measured using Archimedes Principle

-- = Specific Energy measured after chip formation

TABLE C-5 (continued) - Specific Energy Calculations

CONCRETE SPECIMENS - PRODUCTION TESTS									
	Test Series	Impact Set (1)	Weight of Cuttings, g	Weight of Cuttings, ton	Specific Energy Hp-hr/ton	Volume (2) of Crater, cc	Volume (2) of Crater, cu yd	Specific Energy Hp-hr/cu yd	
LOW VELOCITY V = 9.3 ft/sec Cutter Weight = 485 lb Impact Energy = 651 ft-lb (per button) = 217 ft-lb = 1.10E-04 Hp-hr	Low Strength UCS = 3624 psi	RPLW3A1-3	1	23.3	2.6E-05	12.8	---	---	---
		RPLW3A21-3	2	32.5	3.6E-05	10.7	---	---	---
		RPLW3B1-3	1	16.0	1.8E-05	18.6	---	---	---
		RPLW3B21-3	2	366.7	4.0E-04	1.3	---	---	---
		RPLW3C1-3	1	10.3	1.1E-05	29.0	---	---	---
		RPLW3C21-3	2	295.8	3.3E-04	1.3	495	6.5E-04	1.3
	High Strength UCS = 9,916 psi	RPLS3A1-3	1	11.0	1.2E-05	27.1	---	---	---
		RPLS3A21-3	2	33.4	3.7E-05	13.4	---	---	---
		RPLS3A31-3	3	42.5	4.7E-05	10.3	55	7.2E-05	13.7
		RPLS3B1-3	1	25.2	2.8E-05	11.8	---	---	---
		RPLS3B21-3	2	50.2	5.5E-05	7.9	---	---	---
		RPLS3B31-3	3	27.0	3.0E-05	8.7	55	7.2E-05	13.7
		RPLS3C1-3	1	25.7	2.8E-05	11.6	---	---	---
		RPLS3C21-3	2	48.6	5.4E-05	8.0	---	---	---
		RPLS3C31-3	3	33.5	3.7E-05	8.3	50	6.5E-05	15.1
HIGH VELOCITY V = 18.9 ft/sec Cutter Weight = 130 lb. Impact Energy EL = 721 ft-lb (per button) = 240 ft-lb = 1.21E-04 Hp-hr	Low Strength UCS = 3624 psi	RPHW3A1-3	1	24.6	2.7E-05	12.1	---	---	---
		RPHW3A21-3	2	41.7	4.6E-05	9.0	---	---	---
		RPHW3B1-3	1	28.1	3.1E-05	10.6	---	---	---
		RPHW3B21-3	2	364.8	4.0E-04	1.3	---	---	---
		RPHW3C1-3	1	71.3	7.9E-05	4.2	---	---	---
		RPHW3C21-3	2	474.1	5.2E-04	1.1	655	8.6E-04	0.3
	High Strength UCS = 9,916 psi	RPHS3A1-3	1	22.5	2.5E-05	13.3	---	---	---
		RPHS3A21-3	2	35.8	3.9E-05	10.2	---	---	---
		RPHS3A31-3	3	154.6	1.7E-04	4.2	---	---	---
		RPHS3B1-3	1	59.6	6.6E-05	5.0	---	---	---
		RPHS3B21-3	2	50.7	5.6E-05	5.4	---	---	---
		RPHS3B31-3	3	59.6	6.6E-05	5.3	170	2.2E-04	4.4
		RPHS3C1-3	1	72.8	8.0E-05	4.1	---	---	---
		RPHS3C21-3	2	41.2	4.5E-05	5.2	---	---	---
		RPHS3C31-3	3	45.1	5.0E-05	5.6	60	7.8E-05	12.6

(1) 3-inch row spacing for all tests

(2) Volume measured using glycerin

 = Specific Energy measured after chip formation

TABLE C-5 (continued) - Specific Energy Calculations

		ROCK SPECIMENS												
		Test Series	Row Spacing, in	Impact Set	Cutting Gradation (1)	Weight of Cuttings, g	Weight of Cuttings, ton	Specific Energy Hp-hr/ton	Volume (2) of Crater, cc	Volume (2) of Crater, cu yd	Specific Energy Hp-hr/cu yd	Volume (3) of Cuttings, cc	Volume (3) of Cuttings, cu yd	Specific Energy Hp-hr/cu yd
LOW VELOCITY V = 9.8 ft/sec Cutter Weight = 485 lb Impact Energy = 723 ft-lb (per button) = 241 ft-lb = 1.12E-04 Hp-hr	Berea Sandstone UCS = 6,700 psi	RPLBSA1-3	0	1	--	92.9	1.02E-04	3.6	--	--	--	--	--	--
		RPLBSA21-3	0	2	--	47.1	5.19E-05	4.7	--	--	--	--	--	--
		RPLBSB1-3	3	1	--	69.0	7.61E-05	4.8	--	--	--	--	--	--
		RPLBSB21-3	3	2	--	197.9	2.18E-04	2.5	155	2.0E-04	3.6	--	--	--
	Barre Granite UCS = 28,600 psi								--	--	--	--	--	--
		RDLBGA1-3	0	1	WG	9.2	1.01E-05	36.0	--	--	--	--	--	--
		RDLBGA21-3	0	2	--	16.7	1.84E-05	25.6	--	--	--	--	--	--
		RDLBGA31-3	0	3	WG	12.0	1.32E-05	26.2	34	4.4E-05	24.6	15.6	2.0E-05	53.7
		RDLBGB1-3	2	1	--	20.1	2.22E-05	16.5	--	--	--	--	--	--
		RDLBGB21-3	2	2	--	20.1	2.22E-05	16.5	--	--	--	--	--	--
		RDLBGB31-3	2	3	--	6.4	7.06E-06	21.3	32	4.2E-05	26.2	--	--	--
		RPLBGA1-3	0	1	--	15.8	1.74E-05	21.0	--	--	--	--	--	--
		RPLBGA21-3	0	2	--	24.0	2.65E-05	16.6	--	--	--	--	--	--
		RPLBGA31-3	0	3	--	12.0	1.32E-05	19.2	29	3.8E-05	28.9	--	--	--
		RPLBGB1-3	3	1	--	20.2	2.23E-05	16.4	--	--	--	--	--	--
		RPLBGB21-3	3	2	--	11.3	1.25E-05	21.0	--	--	--	--	--	--
		RPLBGB31-3	3	3	--	9.0	9.92E-06	24.5	21	2.7E-05	39.9	--	--	--
		RPLBGC1-3	1.5	1	--	46.5	5.13E-05	7.1	--	--	--	--	--	--
		RPLBGC21-3	1.5	2	--	41.3	4.55E-05	7.5	--	--	--	--	--	--
		RPLBGC31-3	1.5	3	--	54.0	5.95E-05	7.0	--	--	--	--	--	--
									--	--	--	--	--	--

(1) WG = Well Graded, GG = Gap Graded, PG = Poorly Graded

(2) Volume measured using glycerin

(3) Volume measured using Archimedes Principle

= Specific Energy measured after chip formation

RI 9374

REPORT OF INVESTIGATIONS/1991

Testing of a Diesel-Powered Impact Cutting Head for Hard-Rock Mining

By P. D. Kavscek, C. D. Taylor, and H. Handewith

UNITED STATES DEPARTMENT OF THE INTERIOR

BUREAU OF MINES

Mission: As the Nation's principal conservation agency, the Department of the Interior has responsibility for most of our nationally-owned public lands and natural and cultural resources. This includes fostering wise use of our land and water resources, protecting our fish and wildlife, preserving the environmental and cultural values of our national parks and historical places, and providing for the enjoyment of life through outdoor recreation. The Department assesses our energy and mineral resources and works to assure that their development is in the best interests of all our people. The Department also promotes the goals of the Take Pride in America campaign by encouraging stewardship and citizen responsibility for the public lands and promoting citizen participation in their care. The Department also has a major responsibility for American Indian reservation communities and for people who live in Island Territories under U.S. Administration.

Report of Investigations 9374

Testing of a Diesel-Powered Impact Cutting Head for Hard-Rock Mining

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UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

ft	foot	kHz	kilohertz
ft/s	foot per second	lb	pound
Gal	acceleration = 386 in/s^2	lb/ft ³	pound per cubic foot
hp	horsepower	psi	pound (force) per square inch
hp·h/st	horsepower hour per short ton	s	second
Hz	hertz	st/h	short ton per hour
in	inch	yd ³	cubic yard

TESTING OF A DIESEL-POWERED IMPACT CUTTING HEAD FOR HARD-ROCK MINING

By P. D. Kavscek,¹ C. D. Taylor,² and H. Handewith³

ABSTRACT

The performance of a novel prototype kerf-cutting impact mining machine was evaluated under a cooperative agreement between the U.S. Bureau of Mines and RAMEX Systems, Bellevue, WA, while operating under conditions typical of normal tunnel entry development. Selected operating parameters were monitored concurrently to determine baseline operating conditions and to study relationships between operating parameters. Using the data obtained, the specific energy requirements of the impact mining machine were calculated and compared to specific energy requirements of tunnel boring machines cutting in rock having similar hardness. Tests results indicate that the kerf-cutting impact mining machine can provide a mechanical means for mining very hard rock that cannot be effectively mined using commercially available mechanical excavators.

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INTRODUCTION

To increase productivity, the mining industry must sometimes try innovative types of mechanized mining systems. In the past, much of this technology has been based on the use of mining machines, such as the road-header, that use rotary cutting heads with drag bits. However, the effectiveness of these machines is limited by rock hardness. If the rock is too hard to be cut by a mining machine, then explosives may be used. Mining with explosives is a labor-intensive operation, and the fracturing and weakening of surrounding rock strata are difficult to control.

Other types of mining machines that can cut in some harder rock use disk cutting bits. However, most of these machines, such as the tunnel borer, are large, difficult to maneuver and limited to cutting entries having a uniform diameter.

A new type of mechanized mining machine has been developed to fill the gap in technology between mechanical cutting machines and explosives or tunnel boring machines. Energy is delivered to an impact cutting head, eight to nine times per second, by a single free-floating piston. With an impact velocity of 10 ft/s, the cutting head is capable of fracturing and creating kerfs in rock too hard for most mechanical machines to cut. The machine can be designed for easy maneuvering in an underground mine,

and its articulated boom allows the cutting of entries of varying size.

The prototype kerf-cutting impact mining machine was built by RAMEX Systems of Bellevue, WA.⁴ For the initial tests, the impact mining boom of the machine was mounted on the body of a diesel-powered excavator (fig. 1). The mining boom can be mounted on a conventional backhoe undercarriage for some demolition, secondary breaking, and tunneling operations (fig. 2). For underground use, the boom can be adapted to various types of undercarriages. Some underground mining operations require an undercarriage that can gather, convey, and load the rock that is removed. The mining machine can be made small enough so that it can be easily maneuvered in underground entries and taken in and out of the mine.

RAMEX Systems and the U.S. Bureau of Mines entered into a cooperative agreement to evaluate the performance of this mining machine. RAMEX Systems operated the prototype machine in a rock quarry, and the Bureau provided equipment and personnel for monitoring machine operating parameters.

⁴Handewith, H. J., W. D. Coski, and E. D. Thimons. Development of a Prototype Hard Rock Excavating Machine. Paper in Proceedings of 1989 Rapid Excavation and Tunneling Conference Los Angeles, CA, June 11-14, 1989, SME, 1989, pp. 769-787.

Figure 1.—Impact mining machine.

Figure 2.—Impact breaker performing tunneling operation.

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mechanic; and Zepu Liang, test engineer. Allen Constantine, electronics technician, Schneider Services, Pittsburgh, PA assisted in the preparation and operation of the data collection system.

DESCRIPTION OF MACHINE DESIGN AND FUNCTION

Figure 3 shows the main internal components of the mining boom, which is called the Ram. A two-stroke, single-cylinder diesel engine moves a 700-lb piston inside the cylinder. The piston is free floating. To initially compress the air and attain fuel ignition, compressed air is released suddenly through a valve in the cylinder bounce chamber. Just before the piston reaches the combustion head, a linear cam shaft attached to the piston opens the fuel line valve, allowing fuel to pass through the diesel injector. The hot compressed air ignites the fuel. The combustion pressure stops the piston and causes it to move back toward the bounce chamber.

Compressed air in the bounce chamber stops the piston before it reaches the end of the cylinder. Without metal-to-metal contact, energy is transferred through the cushion of air to the cutting head. This cushion of compressed air

also returns the piston to the combustion end of the cylinder where the cycle is repeated.

The cutting head (fig. 4) is specially designed to take advantage of the energy transferred from the piston. The 16- by 16-in head, which is bolted to the forward end of the Ram, has seven rows of $3/4$ in tungsten carbide buttons that are spaced 2 in apart. Upon impact, with this spacing design, rock breaks out between the rows of buttons and creates a kerf or slot (fig. 5).

To develop a tunnel entry, several kerfs must be cut in the face. The cutter starts by sumping in about an inch deep at the crown and then trams down the tunnel face. Trammings the boom up and down produces a deep slot with five kerf-cut mounds on the bottom. The slot is deepened to about 1 ft by repeated trammings. The number of passes and depth of slot are determined by the

Figure 3.—Main components of Ram.

Figure 4.—Ram cutting head.

hardness of the rock formation being excavated. The tunnel gage is outlined with slots cut in this manner. Additional slots are then cut at convenient spacing across the tunnel face. The core or mound left between the slots, being unconfined on three sides, is broken out by the impacts of the cutting head.

As a new rock surface is exposed, subsequent impacts create new kerfs, and the depth of cut increases incrementally with each impact. The ability to control the shape and depth of the excavation is one of the primary

Figure 5.—Rock fracture pattern with Ram cutting head.

advantages of using an impact mining machine with a kerf-cutting head.

Thrust force and Ram position are manually controlled by the operator. A steady thrust force is applied to the reciprocating Ram through a hydraulically actuated spring mechanism (fig. 3). The hydraulic mechanism senses thrust pressure and is used to automatically regulate engine throttle setting according to the power demand. This system also prevents operation of the engine when there is no power demand. Without this feature, destructive internal forces could occur.

EXPERIMENTAL PROCEDURES

Field-testing was conducted at a pit quarry site near Monroe, WA. The test rock was a massively jointed felsitic andesite that occurred at the quarry in a flow about 40 ft thick. The volcanic intrusive rock has a compressive strength of about 40,000 psi.

The machine was located at the base of a vertical 45-ft highwall. A wire mesh was bolted to the top of the bench and stretched above the test site to prevent rockfalls. During testing, the machine began developing a 10- by 10-ft simulated mine entry. For each test, the cutting boom was moved so that an 8-ft vertical slot was cut. Width of the slot was the same as the cutting head (16 in). The cutting boom, which was usually positioned horizontally to begin a test, was trammed up and down so that the same slot depth was maintained for the full length of cut.

Instrumentation for monitoring operating parameters was situated in a trailer located adjacent to the test site. Signal cables were extended from the trailer to the transducers, which monitored operating parameters on the mining machine and ground surface near the test site.

A block diagram of the data collection system is shown in figure 6. The type of transducers used and the parameters measured are discussed in the following section.

MACHINE PARAMETERS MONITORED

Piston and Ram Displacement

A pinion gear, attached to the shaft of a rotary potentiometer (0 to 500 ohm), was turned by a rack that was connected to the piston. A constant voltage was supplied to the potentiometer. The potentiometer was calibrated so that the voltage drop across the potentiometer was related to the piston displacement in the cylinder.

Ram movement, relative to the machine support structure, was monitored using a 12-in linear potentiometer. The upper end of the potentiometer was attached to the upper end of the Ram; the lower end of the transducer was attached to the machine support structure (fig. 7). A constant voltage was supplied to the potentiometer, which was calibrated so that the voltage drop across the potentiometer could be related to the Ram displacement relative to the machine support structure.

Combustion Pressure

A 0- to 5,000-psi, quartz-type pressure transducer was placed in the combustion chamber to monitor combustion pressure during each engine cycle.

Cutter Head Acceleration

A 0- to 200-Gal strain gauge accelerometer was installed on the top surface of the cutter head (fig. 8) to monitor the force delivered to the head in a direction parallel to the axis of the Ram (i.e., perpendicular to the rock surface).

Machine Body Force Measurements

Quartz-type accelerometers (0 to 50 Gal) were installed on the machine chassis beneath the operator's station and adjacent to the attachment point of the mining boom (fig. 9). Forces on the machine chassis were measured in three orthogonal directions (x = horizontal plane, y = vertical plane, and z = right or left, perpendicular to the axis of the Ram).

Rock Motion

High-output, low-distortion, 10-Hz geophones were installed approximately 25 ft from the cutter head at locations on the quarry highwall face and bench. Ground motion in the vertical direction was monitored at both locations.

Two quartz-type accelerometers (0 to 50 Gal) were installed on the highwall. One accelerometer, located 12 ft from the cutter head, measured rock motion in the horizontal direction. The second accelerometer, which was positioned adjacent to the highwall geophone, measured rock motion in the vertical direction.

Signal Conditioning and Recording

Charge amplifiers were used to power and condition the sensor signals from the combustion pressure transducer. Signals from the linear transducer were conditioned by instrumentation amplifiers. A full-bridge amplifier was used to power the sensor and condition the signal from the cutter head transducer. Battery-powered amplifiers were used to supply power and condition the sensor signals from the machine- and ground-mounted accelerometers. The geophone output required no signal conditioning and was connected directly to the magnetic tape recorder.

A 14-channel Racal⁵ frequency modulated (FM) magnetic tape recorder, configured for a 10-kHz frequency

⁵Reference to specific products does not imply endorsement by the U.S. Bureau of Mines.

Figure 6.—Data collection system.

Figure 7.—Linear potentiometer for Ram displacement.

Figure 8.—Cutting head accelerometer.

Figure 9.—Machine body accelerometers.

response, was used to record the machine and motion data. The magnetic tape enabled high-speed concurrent recording of data collected by the 14 sensors. Subsequent analysis of the data was performed by playing back the information through the recorder.

An oscilloscope was used at the test site to monitor and verify the tape recorder signals (fig. 10). Hard copies of the data were plotted at the test site using a fiber optics chart recorder.

DATA ANALYSIS

Plots Of Operating Parameters

Plots made with the fiber optics recorder were used to make a cursory inspection of all the data obtained. Approximately 5 s of data from one test segment was selected for a more detailed analysis. These data were replayed into an analog-to-digital converter (using Honeywell H-TMS 3000 instrumentation). Each channel was sampled at a rate of 2,400 samples per second. The analog-to-digital sample rate was selected to obtain

Figure 10.—Data Instrumentation.

accurate reproduction of the recorded information in both amplitude and frequency.

The digitized data were stored as American National Standard Code for Information Interchange (ASCII) files and plotted using the Lotus 1-2-3 software program. The type of plot obtained for each of the parameters monitored is shown in figures 11 through 16. Data included in each plot were obtained during the same period and represent about 0.2 s of testing time. This corresponds to approximately one complete engine cycle.

The plot in figure 11 illustrates how the piston moves between the combustion and bounce chambers. The total distance of piston movement is about 10 in. Fuel ignition occurs once during each cycle of piston movement. The height of the spike (fig. 12) resulting from each ignition shows the combustion pressure produced.

Following combustion, the Ram moves in a direction away from the cutting surface (fig. 13). For the engine cycle plotted, the Ram moved approximately 0.6 in before the constant thrust of the hydraulically actuated springs reversed the direction of motion. After impact, the Ram cutting head penetrated approximately 0.1 in of the rock. After initial impact, the cutting head rebounded from the rock and bounced several more times. No significant additional rock penetration occurred.

Force levels measured by the accelerometers placed on the cutting head, machine body, and ground are seen in figures 14, 15, and 16. As expected, the highest force level was recorded by the cutting head accelerometer after the initial rock impact (fig. 14).

Force levels on the machine body are shown for the x-direction (fig. 15). Although smaller in magnitude than the force levels measured on the cutting head, a significant amount of energy was transmitted back through the machine body.

A plot of ground force is given for the accelerometer position on the highwall, approximately 25 ft from the test cutting location (fig. 16). Force levels measured at all the ground sensing locations were less than on the cutting head or machine body. All ground sensing locations recorded force levels corresponding to the initial and secondary impacts.

The information in figures 11 through 16 represents data that were collected at the same time; therefore, the plots can be easily combined on the coinciding time bases (x-axis) to better show the relationship between the operating parameters. For example, figure 17 includes information for piston position and combustion pressure. Fuel ignition occurred just before the piston reached the combustion end of the cylinder.

The correlation between Ram movement and transfer of energy to the cutting head is shown in figure 18. The highest force level is related to the initial contact between

Figure 11.—Piston movement.

Figure 12.—Combustion pressure.

Figure 13.—Ram displacement.

Figure 14.—Force levels measured on cutting head.

Figure 15.—Force levels measured on machine body.

Figure 16.—Force levels measured on ground.

Figure 17.—Piston displacement and combustion pressure.

Figure 18.—Ram displacement and cutter head force.

the cutting head and rock. The head rebounded from the rock and subsequently struck it several more times. The additional strikes transferred small amounts of energy to the rock, but as previously noted, no significant additional rock penetration resulted.

Calculation Of Specific Energy

The specific energy of a mining machine is the energy required to remove a given volume of material. It is a function of many factors including cutting tool force and geometry, and rock strength. The cutting efficiencies of mining machines may be compared by calculating specific energies for machines operating under similar conditions. To calculate specific energy, the energy supplied by the mining machine and the volume removal rate must be known.

Calculation Of Energy Input

Operating power level of the diesel engine, rather than the rated horsepower, represents the energy that is actually delivered to the cutter head. The operating power level for the kerf-cutting impact mining machine was calculated using the combustion pressure and the piston location measurements. Summing the integrals:

$$\text{Impact energy} = \int_{\text{TDC}}^{\text{BDC}} F(x)Ddx + \int_{\text{BDC}}^{\text{TDC}} F(x)Ddx,$$

where $F(x)$ = combustion pressure at piston location x ,

D = bore diameter,

BDC = bottom dead center,

X = piston location,

and TDC = Top dead center

gives a calculation of the energy delivered to the rock with each impact. The shaded area in figure 19, which shows combustion pressure versus piston location, is the energy for one cycle. The power output was then calculated as the change in energy with respect to time. The average power delivered to the cutter head was between 35 and 40 hp.

The average energy supplied for several consecutive engine cycles can also be calculated. Figure 20 shows the energy for six cycles during a 0.7-s period. The average power supplied during these cycles was 36 hp.

Figure 19.—Combustion pressure versus piston location.

Calculation Of Material Removal Rate

The specific energy (horsepower hour per short ton) is determined by dividing the operating power level (horsepower) by the rate of mining (short tons per hour). Normally, material removal rate is determined by weighing the material removed in a given period or, if the material density is known, by measuring the volume of material

removed in a given time. For the tests conducted, the operation time was not long enough for an accurate determination of material weight or volume.

For these tests, data collected to determine Ram displacement were used to calculate the depth of cut per impact. Total depth of cut for 45 impacts was about 1.8 in (fig. 21). Therefore, the average depth of cut per impact was 0.04 in.

Figure 20.—Energy supplied during six engine cycles.

Figure 21.—Ram displacement during 20 impacts.

Volume removed with each impact was calculated by assuming equal penetration across the entire area of the cutting head (i.e., 16 by 16 in). Density of the quarry rock was taken to be 170 lb/ft³. During testing, the average material removal rate was 14.5 st/h. This accounts only for material removed from beneath the cutting head.

During a typical mining operation, material will also break out from between the kerfs, and the mining rate will be higher. However, based on a removal rate of 14.5 st/h, the specific energy for the 36-hp engine is between 2 and 3 hp·h/st.

DISCUSSION AND CONCLUSIONS

SPECIFIC ENERGY

A key measure of cutting efficiency is the specific energy required for a given operation. The calculated specific energy for the kerf-cutting impact mining machine, mining in the test rock, was 2 to 3 hp·h/st.

It is difficult to relate the performance of this machine to other mining machines because most of those machines use either drag or disk bits that are attached to a rotating cutting head. Machines using drag bits require a specific energy of less than 1 hp·h/st to cut softer, less abrasive rocks, such as coal. Harder rocks, such as those cut by the impact mining machine, cannot be mined using drag bits.

Disk cutters on tunnel boring machines have been used to cut rock of comparable hardness (e.g., greenstone, quartzite, and granite). With the tunnel boring type of

machine, calculated specific energies vary from 10 to 18 hp·h/st. Comparing the specific energy requirements for the kerf-cutting impact miner (i.e., 2-3 hp·h/st) and tunnel boring machines, the impact mining machine appears to provide an efficient means for mining very hard rock.

Theoretical specific energy curves (fig. 22) were drawn for 100, 40, and 36 hp engines. For the 36 hp engine, it can be seen that when the depth of cut is about 0.04 in, the specific energy is between 2 and 3 hp·h/st. For the same horsepower engine, specific energy decreases as the depth of cut increases. A larger engine may be required to increase the depth of cut. To maintain the same cutting efficiency (i.e., specific energy), a 100-hp engine would have to have a penetration rate of greater than 0.1 in per impact.

Figure 22.—Theoretical specific energy curves.

HORSEPOWER RATING OF DIESEL ENGINE

The diesel engine used for these tests was rated at 100 hp. The calculated power delivered to the cutting head was 36 hp. The primary reason the rated and delivered horsepower differed during the testing was that, due to a fuel injection system problem, the machine was not operating at full capacity. Since the testing, improvements have been made to the injection system and higher power outputs recorded.

EFFICIENCY OF CONVERSION—COMBUSTION TO RAM MOVEMENT

The data plotted in figures 11 and 13 show, qualitatively, how the energy from fuel combustion was converted to piston and Ram movement. The efficiency of this transfer was not calculated. However, unlike a hydraulic breaker, there are no hydraulic losses, and the bulk of the combustion energy is transferred into rock breaking. The only loss is heat transferred through the combustion chamber and sliding mechanical friction of the piston on the cylinder wall. Thus, the efficiency of the energy transfer is considered high.

ENERGY TRANSFERRED THROUGH MINING MACHINE

Not all of the energy generated by fuel combustion is transmitted through the cutting head. A part of the energy (fig. 15) is transmitted back through the machine body. This energy is wasted and potentially damaging to the mining machine body. Further study is needed to determine if the force levels measured can cause damage to the machine. Reducing the energy transferred through the machine body could increase machine life and, possibly, increase the energy transferred through the cutting head.

ENERGY TRANSFERRED THROUGH ROCK

With the Ram, rock fracture results when energy is transferred from the cutting head to the rock. During the cutting tests, ground motion was monitored in an attempt to determine how the energy directed to the rock affects the amount of rock fracture.

It was not possible to measure the force delivered to the rock at the location of cutting head impact. Rather, the geophones and accelerometers were attached to the rock, at locations near where the head struck the rock, but far enough away so that they were not damaged by the impacts.

At the sensor locations, the force levels measured were much less than force levels on the cutting head (figs. 14 and 16). The force-induced vibrations decreased as they traveled through the rock to the sensor locations due to several factors, including

1. Distance and direction between the location of impact and point of measurement: The ground accelerometer, which monitored acceleration in the vertical direction, was placed about 25 ft from the location of impact.
2. Material composition and discontinuities in rock material: The quarry rock was composed of nonhomogeneous rock that in many locations was fractured or contained thin veins of unconsolidated material.

During the testing, it was not possible to interpret how much the force levels had been reduced due to the distance between the sensors and location of impact, and the material composition. Further work is needed to evaluate how the magnitude of the force levels delivered to the rock surface are related to amount of rock fracturing that results.

ENERGY TRANSFER—ROCK FRACTURE AND PRODUCT SIZE

One way to determine how efficiently energy is transferred from the cutting head of the mining machine to the rock is to look at the size of the fractured material. The mining operation and machine are designed to produce a certain-sized product. Producing a product smaller than the desired size wastes energy.

The cutting head of the impact mining machine was designed to "break out" oyster-shell-sized rock chips when the head impacts the rock. To prevent additional fracturing of these chips, they must immediately be removed from under the cutting head before the next impact. A chip that does not fall out immediately will be crushed by the next impact, and part of the energy will be wasted by the additional fracturing. In addition, the fractured material will form a cushion between the tungsten carbide buttons and the solid rock, thus making additional chip formation more difficult. Removal of the chips allows the tungsten buttons to come in direct contact with the solid rock.

The rock cuttings produced during these tests varied in size from plus 10 in down to dust. Techniques for more efficiently removing the rock chips from beneath the cutting head have been investigated. During the test program, compressed air was directed through four holes that were drilled through the bottom of the cutting head. The

holes were placed so that the air passed between the tungsten carbide buttons. Part of the material removed by the compressed air was finely crushed rock. This resulted in increased concentrations of airborne dust near the cutting boom.

Future testing of the impact mining machine will include use of high-pressure jets of water that will be directed between the tungsten carbide bits. Operating at a pressure of about 10,000 psi, the water jets should "force out" the rock chips and reduce dust resulting from secondary crushing.

The test program conducted has shown that the kerf-cutting impact mining machine has the potential to mine rock materials that are too hard to be cut efficiently with currently available mechanized mining machines. Moreover, the impact mining energy requirements are less than for other commercially available mechanical excavators cutting in rock of similar hardness.

OPERATING COSTS

Operating costs are dependent on the life of the cutting bits, muck gathering and conveying, and other wear components. A computer program was developed to compare operating costs of the RAMEX mining boom, drill-blast techniques, and other mining methods.⁶ Results of this program indicated that a mining machine with a 400-hp RAMEX boom could be cost and performance competitive with explosive excavations in many of the same rock formations.

During the test program described in this report, only about 80 st (35.9 yd³) of material were mined. This amount of mining resulted in little noticeable wear on the cutting bits. A longer test period with a full-scale mining machine is needed to evaluate the costs associated with the wear of cutting bits and other machine components.

⁶Work cited in footnote 4.