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Dominant rock properties affecting the performance of conical picks and the comparison of some experimental and theoretical results

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Abstract

Conical picks are the essential cutting tools used especially on roadheaders, continuous miners and shearers and their cutting performance affects directly the efficiency and the cost of rock/mineral excavation. In this study, in order to better understand the effects of dominant rock properties on cutter performance, 22 different rock specimens having compressive strength values varying from 10 to 170 MPa are first subjected to a wide range of mechanical tests. Then, laboratory full-scale linear cutting tests with different depth of cut and cutter spacing values are realized on large blocks of rock specimens using one type of conical pick. Specific energy, cutting and normal force values for relieved and unrelieved cutting modes are recorded using a triaxial force dynamometer with capacity of 50 tonnes and a data acquisition system. Cutter force and specific energy values are correlated with rock properties and theoretical force and specific energy values obtained from widely used theoretical approaches.

The results indicate that uniaxial compressive strength among the rock properties investigated is best correlated with the measured cutter performance values, which is in good agreement with previous studies. However, it is also emphasized in this study that Brazilian tensile strength, Schmidt hammer rebound values, static and dynamic elasticity modulus are also dominant rock properties affecting cutter performance.

Theoretical specific energy defined by different researchers has a meaningful relationship with the experimental specific energy, which is an essential parameter for predicting the instantaneous cutting rates of mechanical excavation systems. It is also demonstrated that the experimental cutter forces obtained for 5 mm depth of cut are in good agreement with theoretical force values, if the friction angle between rock and cutting tool is included in the theoretical formulation. It is emphasized that, to some extend, laboratory tests can help to minimize high cost of a trial–error approach in the field.

1. Introduction

Application of excavation machines for hard rock excavation in both civil and mining engineering fields has increased significantly in recent years and a full-scale laboratory cutting test has emerged as a necessity to provide basic data for machine selection, design and performance prediction for a given rock formation [1–5].

The ability of excavation machines to operate and cut effectively in hard rock is limited by the system stiffness and the ability of cutting tools to withstand high forces. Mean and peak cutter forces, which are obtained with high reliability from the full-scale linear cutting tests, are of vital importance for a given rock formation. The force acting on a cutting tool changes constantly in magnitude during a cutting process due to chipping and brittle nature of the rock. Averages of all the force changes during the course of cutting action gives mean cutter forces and mean peak forces are average of the peak forces for a given cutting condition. High forces may result in gross fracture damage to the tungsten

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carbide cutting tip, damage the machine components and exceed the machine's torque and thrust capacities. Therefore, it is essential to understand the basic aspects of rock cutting mechanics to minimize the large cost of a trial and error approach with an excavation machine in the field.

A number of researchers have studied the theoretical aspects of coal and rock cutting process for the last 40 years. However, the most comprehensive and accepted theories are those of Evans' [6–10] for chisel picks and conical picks and of Nishimatsu's [11] for chisel picks. Although these theories led to a better understanding of the coal and rock cutting process, the detailed laboratory cutting experiments have been always felt a necessity, since in some cases theoretical estimations of cutter forces were not in good agreement with actual cutting forces due to the complex petrographic, mineralogical and anisotropic nature of rock formations [12].

Determination of dominant rock properties affecting the performance of conical picks and establishing reliable equations for performance prediction are the main objectives of this paper. The comparison of theoretical and experimental results is also one of the main objectives. An extensive engineering research programme on rock excavation is established in the Mining Engineering Department of Istanbul Technical University using 22 different large rock specimens in which the rock cutting mechanics are largely evaluated experimentally.

2. Previous studies, theoretical and practical considerations

2.1. Theoretical and experimental studies on cutting forces

A number of scientists have formulated mathematical models to improve the design of the excavation machines and find the best configuration of the cutting tools for more efficient cutting process [13]. The pioneering work on coal cutting mechanics performed by Evans [6] and Evans and Pomeroy [14] and extended theoretical works of Evans [7–10] were used to establish the basic principles of the cutting process and these have been widely used in the efficient design of excavation machines such as shearers, continuous miners and roadheaders [13]. Evans demonstrated theoretically that tensile strength and compressive strength were dominant rock properties in rock cutting with chisel picks and point attack tools as formulated below in Eqs. (1) and (2)

$$\mathrm{FC} = \frac{2\sigma_t d\, w \operatorname{Sin}\frac{1}{2}\left(\frac{\pi}{2}-\alpha\right)}{1-\operatorname{Sin}\frac{1}{2}\left(\frac{\pi}{2}-\alpha\right)} \quad \text{for chisel picks,} \tag{1}$$

$$\mathrm{FC} = \frac{16\pi d^2\sigma_t^2}{\operatorname{Cos}^2(\varphi/2)\sigma_C} \quad \text{for point attack tools,} \tag{2}$$

where FC is cutting force, d is depth of cut, w is tool width, α is rake angle, σ_t is tensile strength, σ_C is compressive strength and φ is tip angle. He also formulated optimum spacing for chisel picks as three to four times the pick width [8].

Roxborough proved that the experimental forces for chisel picks were in good agreement with theoretical values calculated using Eq. (1) [15]. Ranman described also a rock cutting model for conical tools [16]. Guo and co-workers showed that, compared with conventional rock mechanics methods such as well-known Mohr–Coulomb failure criterion, linear elastic fracture mechanics could furnish a greater insight to the rock cutting mechanics [17]. The fracture mechanics approach provided detailed information on crack progressive failure, crack propagation path, corresponding load requirement and stability of the crack propagation.

Goktan suggested a modification on Evans' cutting theory for point attack tools as indicated in Eq. (3) and concluded that the force values obtained with this equation were close to previously published experimental values and could be of practical value, if confirmed by additional studies [18].

$$\mathrm{FC} = \frac{4\pi\sigma_t d^2 \operatorname{Sin}^2(\varphi/2+\psi)}{\operatorname{Cos}(\varphi/2+\psi)}, \tag{3}$$

where ψ is friction coefficient between cutting tool and rock and other parameters as defined above for Eqs. (1) and (2).

Roxborough and Liu suggested also a modification on Evans' cutting theory for point attack tools as indicated in Eq. (4). They concluded that for Grindleford sandstone the predicted mean peak cutting force values are in good agreement with the modified cutting theory. However, the friction angle used was found as 16° using a steel block and a natural flat rock surface [19].

$$\mathrm{FC} = \frac{16\pi\, \sigma_C d^2\sigma_t^2}{\left[2\sigma_t + (\sigma_C \operatorname{Cos}(\varphi/2))\left(\frac{1+\operatorname{Tan}\psi}{\operatorname{Tan}(\psi/2)}\right)\right]^2}. \tag{4}$$

The notation in Eq. (4) is as explained before.

Goktan used Evans' theories to compare the cutting efficiency of point attack tools and wedge–shaped picks and concluded that the ratio of tensile to compressive strength was the main parameter governing the relative efficiency [20]. Goktan developed also some empirical equations to predict the cutting forces of wedge-type cutters and studied the effect of rake angle on the failure pattern of high strength rocks [21,22].

Nishimatsu found that shear strength failure was dominant in cutting high strength rocks as formulated below in Eq. (5) [11]:

$$FC = \frac{2\sigma_S d\, w \operatorname{Cos}(\psi - \alpha) \operatorname{Cos}(i)}{(n+1)[1 - \operatorname{Sin}(i + \psi - \alpha)]} \tag{5}$$

with the additional notations to Eqs. (1) and (2), σ_S is rock shear strength, i is rock internal friction angle and n is stress factor where $n = 12–(\alpha/5)$.

Optimum cutting head and drum design for roadheaders, continuous miners and shearers are recognized as a vital factor in determining the excavation efficiency. In the past, considerable amount of laboratory and in situ research works were carried out in this respect on rock and coal excavation. Hekimoglu and Fowell stated that harmful vibration of the cutting head might be eliminated by the proper design of cutting head; in this context, tilt angles of 65–70° offered the lower specific energy and relative freedom from vibration problems [23,24]. The pick cutters located at the nose portion of the boom-type tunneling machine heads were very often subjected to damage due to skew cutting; therefore, a skew angles had to be provided for the picks with reasonable tilt angles [25]. Skew and tilt angle and other relevant parameters are illustrated in Fig. 1.

Underground trials indicated that the cutting head balance was likely to improve with equally arranged circumferential spacing and equally disturbed pick forces [26]. Hekimoglu and co-workers strongly emphasized that circumferential pick spacing had a remarkable influence on the vibration level of shearer drums, but the concept has not been adequately taken into account in the lacing pattern of some current drums and the cutting performance of drums might be increased considerably if this concept was included in the design [27,28].

Deliac using his extensive theoretical and experimental studies claimed that a new theory based on a simple analysis of the work delivered by a pick during a drum revolution seemed to hold promising results in understanding the behavior of a drum-type cutting head [29]. Furthermore, Deliac in his detailed chapter on "Theoretical and Practical Rules for Mechanical Rock Excavation" reviewed studies on modeling of tool–rock interaction, modeling of rock cutting heads, validation of theoretical models and machine simulation [30].

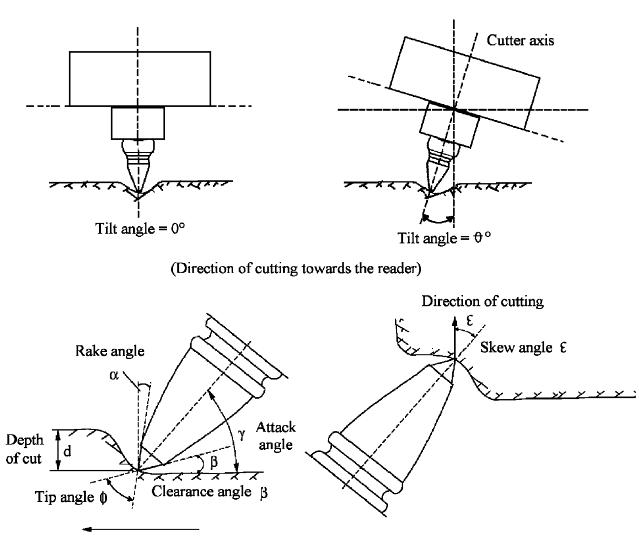

Fig. 1. Some parameters affecting cutting performance of conical tools.

A detailed research study was carried out by Bilgin and co-workers using numerical modeling software and a small-scale rock cutting rig to investigate the effect of lateral stresses on the cutting efficiency of chisel-type cutters [31]. Numerical modeling showed that the lateral stresses dramatically decreased the tensile stresses around the cutting groove up to a certain level of lateral stresses for unrelieved cutting mode. In that case, a lateral stress of 1/5 or 1/4 of rock compressive strength in magnitude caused an increase in cutter force compared to the unstressed condition. However, for relieved cutting mode, the effect of lateral stresses was less apparent, causing an increase in cutter force around 20–30% more than the unstressed conditions. Experimental cutting tests justified the findings of numerical modeling. Those results emphasized that specific energy values found in full-scale relieved cutting tests had to be multiplied by a factor of at least 1.3 in performance prediction models, if tunnels under stress were considered [31].

General principles of efficient cutting head design to increase excavation productivity with less cutting head vibration and less wear of cutting components were investigated in detail by Hurt, MacAndrew and Morris in previous National Coal Board, Mining and Research and Development Establishment (MRDE) [32–36]. They strongly emphasized that cutter force estimation was the essential part of an efficient cutting head design. Later works on rock cutting mechanics in MRDE were mainly concentrated on the cutting performance of point attack tools [37–41]. The results obtained might be summarized as follows: The sharp point attack tools generated higher forces than wedge tools. In abrasive rocks, point attack tools last longer than wedges and might resist higher forces. Minimum cutting forces were exhibited by the point attack tool at an attack angle of 50° corresponding to a back clearance angle of 12°. Angles of skew up to 30° had no great effect on the tool forces. However, during cutting, skew angle provided for a relief on one side of cut line reducing pick damage and a rotation of cutter in tool holder leading to uniform blunting, not self-sharpening, and prolonged tool life.

Intensive experimental studies carried out in United States Bureau of Mines supported the results obtained in MRDE and served to create a good basis to develop coal and rock cutting mechanics. Radial bits appeared to facilitate coal cutting in the tensile mode, while point attack bits appeared to fragment the coal with a more complex mode of failure [42]. Depth of cut was found to be the most significant factor affecting specific energy, cutter forces and airborne respirable dust [43]. These were well summarized in Fowell's work published in "Comprehensive Rock Engineering" [44].

Experimental studies carried out by Roxborough and co-workers [45,46] in order to evaluate some coal cutting theories for continuous miners proved that the normal and cutting forces acting on a cutter increased linearly with depth of cut. Pick spacing had to be considered relative to depth of cut, the chisel-shaped picks were more efficient than the pointed shape tools only at relatively shallow depths of cut. The pointed pick was proved to be consistently more efficient shape at comparatively large depths. There was no evidence to suggest that pick speed had an effect on cutting forces and energies [45,46].

2.2. Theoretical and practical studies on specific energy

Specific energy is one of the most important factors in determining the efficiency of cutting systems and defined as the work to excavate a unit volume of rock. Hughes [47] and Mellor [48] demonstrated that specific energy might be formulated as in the following Eq. (6):

$$\mathrm{SE} = \frac{\sigma_c^2}{2E}, \tag{6}$$

where, SE is specific energy, E is secant elasticity modulus from zero to load to failure and σ_c is compressive strength of rock.

Detailed rock cutting tests, however, showed that specific energy was not only a function of rock properties but it was also closely related to operational parameters such as rotational speed, cutting power of excavation machines and tool geometry. Roxborough reported that specific energy decreased dramatically to a certain level with increasing depth of cut and decreasing tool angle [49–52].

The effect of the spacing between cuts and depth of cut (or penetration) on cutting efficiency is explained in Fig. 2. If the line spacing is too close (a), the cutting is not efficient because the rock is over-crushed; in this region, tool wear is also high due to the high friction between tool and rock, which was well demonstrated in Johson and Fowell's work [53]. They found that tool consumption decreased with available arching force (boom force to penetrate the rock). This was explained as insufficient penetration resulting in rubbing and inefficient unrelieved cutting, which in turn increased tool consumption significantly. This was also found to be true for drilling operation demonstrating that insufficient thrust usually increased tool consumption [54]. Referring to Fig. 2, if the line spacing is too wide (c), the cutting is not efficient since the cuts cannot generate relieved cuts (tensile fractures from adjacent

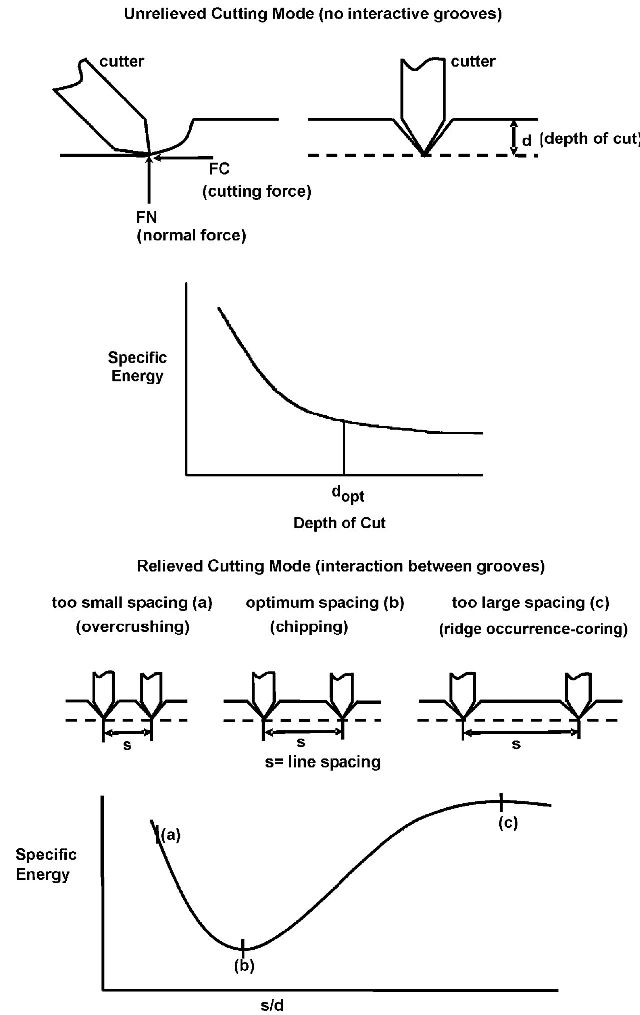

Fig. 2. General effect of cutter spacing on specific energy.

cuts cannot reach to form a chip), creating a groove-deepening situation which creates shock loads causing gross failures in cutting edge. The minimum specific energy is obtained with an appropriate spacing to depth of cut ratio (b). The optimum ratio of cutter spacing to depth of cut generally varies between 1 and 5 for pick cutters.

It is also a fact that cutting modes of roadheaders affect the in situ specific energy values. McFeat-Smith defined four distinct cutting modes for roadheaders: sumping, undercutting, traverse cutting and over-cutting [55]. Fowell and McFeat-Smith noticed that sumping was an inefficient method of excavation, requiring very much higher specific energies than traversing [56]. These modes of cutting were related linearly by the following Eq. (7):

$$\mathrm{SE}_{\mathrm{sumping}} = 3\mathrm{SE}_{\mathrm{traversing}} \tag{7}$$

where, $\mathrm{SE}_{\mathrm{sumping}}$ and $\mathrm{SE}_{\mathrm{traversing}}$ are in situ specific energy requirements during sumping and traversing, respectively.

2.3. Estimating machine cutting rate from specific energy and rock properties

Farmer and Garrity [57] and Pool [58] showed that for a given cutting power of roadheader, instantaneous excavation rate in m^3/h might be predicted by using specific energy values as given in Eq. (6). It is interesting to note also that Krupa, Sekula and co-workers [59–62] noticed that advance rate of a tunnel boring machine for a given power was directly related to specific energy values as formulated in Eq. (6). Kahraman proved also that specific energy values calculated from Eq. (6) might be used as a guide in estimating penetration rates of percussive drills [63].

One of the pioneering works in estimating the performance of roadheaders from specific energy was carried out by McFeat-Smith and Fowell [56]. They measured in situ specific energy by the aid of a Bretby Power Transducer. They reported that cutting rate of roadheaders increased with decreasing in situ specific energy values. However, they noticed that geological weakness planes such as joints, bedding planes and fissures affected tremendously the cutting rates. They defined break index as number of weakness intersecting horizontal and vertical scan line per meter and they found out that specific energy decreased with increasing breaking index. Later, Fowell and Johson reported that there was a good correlation between cutting rate of roadheaders and modified Rock Mass Rating of geological formation [64–67].

Widely accepted rock classification and assessment for the performance estimation of roadheaders was based on the specific energy found from core cutting tests [64–66]. The test involved instrumented cutting tests on 76 mm diameter cores at a depth of cut of 5 mm, cutting speed of 150 mm/s with a chisel-shaped tungsten carbide tool having 10% cobalt by weigh, 3.5-μM nominal grain size, rake angle of (-5°), back clearance angle of 5° and tool width of 12.7 mm. Detailed laboratory and in situ investigations carried out by Fowell and McFeat-Smith showed that there was a close relationship between specific energy values obtained from core cutting tests and cutting rates of medium and heavy-weight roadheaders [68,69]. They formulated core cutting specific energy as in Eq. (8):

$$SE = -0.65 + 0.41\mathrm{CI}^2 + 1.81\,k^{1/3} \pm 2.6\ \mathrm{MJ/m^3}, \tag{8}$$

where SE is laboratory specific energy, CI is cone indenter value, and k is plasticity index. They also reported that tool consumption might be predicted from weight loss of cutter used in core cutting test and Cerchar abrasivity tests [53,64,65,68].

A paper published by Fowell, Richardson and Gollick described large instrumented cutter tests conducted on full-scale boom tunneling research rig led to the development of performance prediction models [70].

Rock cuttability classification based on core cutting test was usually criticized as the effect of rock discontinuities was not well reflected in performance prediction. Bilgin and co-workers developed a performance predictor equation based on rock compressive strength and rock quality designation as given in Eqs. (9) and (10) [71,72]:

$$\mathrm{ICR} = 0.28\,P\,(0.974)^{\mathrm{RMCI}}, \tag{9}$$

$$\mathrm{RMCI} = \sigma_c \cdot \left(\frac{\mathrm{RQD}}{100}\right)^{2/3}, \tag{10}$$

where ICR is instantaneous cutting rate of roadheaders in m^3/h, P is power of cutting head in HP, RMCI is rock mass cuttability index, σ_C is uniaxial compressive strength in MPa and RQD is rock quality designation in percent.

Dunn [73] compared the models described by Bilgin [71,72] and Fowell [68,69] in a research work carried out at Kambalda Mine where Voest Alpine AM75 roadheader was used. Two distinct groups of data were evident. The data grouped around Bilgin's line were strongly influenced by the jointing and weakness zones existed in rock mass. The other group of data was on the line produced by Fowell and Mc Feat-Smith and corresponded to areas where less jointing and weakness zones were present [73].

Thuro and Plinninger defined the area under the stress–strain curve as destruction work which had the unit of specific energy and they proved that there was a good statistical relationship between destruction work and drilling rate of drill rigs and cutting rates of different excavation machines such as roadheaders and TBMs [74–76].

One of the most-accepted method to predict the cutting rate of any excavation machine was to use cutting power, specific energy obtained in laboratory from full-scale linear cutting test and energy transfer ratio from cutting head to rock formation as in Eq. (11) [77,78]:

$$\mathrm{ICR} = k\frac{P}{\mathrm{SE_{opt}}}, \tag{11}$$

where ICR is instantaneous cutting rate in m^3/h, k is energy transfer ratio, P is cutting power of cutting head in kW and SE_{opt} is optimum specific energy in kWh/m^3. Ozdemir strongly emphasized that the predicted value of cutting rate was more realistic if specific energy value in Eq. (11) was obtained from full-scale linear cutting tests at optimum conditions using production cutters. Rostami and Ozdemir pointed out that k changed between 0.45 and 0.55 for roadheaders and from 0.85 to 0.90 for TBMs [78]. They also emphasized that the prediction of optimum specific energy from rock properties would be a great help in estimating advance rates of any excavation machine [79]. Schneider also reported that net cutting rate of roadheader might be found dividing the cutting power of the machine by specific energy, which was closely related to compressive strength of the rock [80]. In situ observations of the other practicing engineers clearly demonstrated that the cutting rate of roadheaders was inversely proportional to rock compressive strength [80–84].

Copur and co-workers stated that if the power and weight of the roadheaders were considered together, the relationship between cutting rate and compressive strength was more realistic [85]. They reported that optimum specific energy values obtained from full-scale cutting tests might be predicted from the product of compressive and tensile strength of the rock [86]. They also defined a set of indices based on macro-scale indentation tests for assessment of rock cutting performance. They concluded that the force and brittleness indices were moderately correlated with cutter performance including specific energy and mechanical properties of rocks [87].

3. Physical and mechanical properties of the rocks tested

Twenty-two different rock samples are collected from different mining and tunneling sites and subjected to physical and mechanical tests explained below. The mechanical tests are performed on core samples taken perpendicular to bedding planes, if exists. The results are summarized in Table 1.

3.1. Uniaxial compressive strength test

Uniaxial compression tests are performed on trimmed core samples, which have a diameter of 54 mm and a length-to-diameter ratio of 2. The stress rate is applied within the limits of 0.5–1.0 MPa/s.

3.2. Brazilian tensile strength

Brazilian tensile strength tests are conducted on core samples having a diameter of 54 mm and a length-to-diameter ratio of 1. The tensile load on the specimens is applied continuously at a constant stress rate such that failure would occur within 5 mm of displacement.

3.3. Static elasticity modulus

Tangent Young's Modulus is measured at a stress level equal to 50% of the ultimate uniaxial compressive strength.

3.4. Dynamic properties

P and S-wave velocities are measured on the cored rock samples having a diameter of 54 mm and a length-to-diameter ratio of 2. The ends of the core samples are

Table 1
Physical and mechanical properties of the rocks tested

Rock name	γ (g/cm^3)	UCS$\pm$SD (MPa)	BTS$\pm$SD (MPa)	E_{sta} (GPa)	E_{dyn} (GPa)	N24 T1	L9 T1	N24 T2	L9 T2	N24 T3	L9 T3	CAI	Ψ (°)
High-grade chromite (*)	4.03	32$\pm$4.4	3.7$\pm$0.6	3.5	31.2	34	43	34	44	30	39	2.12	27
Medium-grade chromite (**)	3.39	47$\pm$10.9	4.5$\pm$0.6	2.3	76.4	47	50	47	54	43	45	1.60	27
Low-grade chromite (***)	2.88	46$\pm$7.6	3.7$\pm$0.8	2.9	35.2	45	52	45	55	43	52	2.40	29
Copper ore (yellow)	4.13	33$\pm$2.5	3.4$\pm$0.02	—	42.0	—	—	—	—	—	—	2.80	38
Copper ore (black)	4.07	41$\pm$3.6	5.7$\pm$0.03	—	49.6	—	—	—	—	—	—	3.00	32
Harsburgite	2.65	58$\pm$27.4	5.5$\pm$1.0	2.1	16.1	49	—	—	—	—	—	0.80	25
Serpantinite	2.49	38$\pm$10.1	5.7$\pm$0.5	2.3	13.9	52	61	52	61	52	61	1.00	28
Trona	2.13	30$\pm$0.7	2.2$\pm$0.4	3.4	3.7	51	52	52	54	39	39	—	30
Anhydrite	2.90	82$\pm$6.0	5.5$\pm$0.8	11.0	—	—	—	—	—	—	—	—	—
Selestite	3.97	29$\pm$2.9	4.0$\pm$0.5	—	—	—	—	—	—	—	—	—	—
Jips	2.32	33$\pm$2.2	3.0$\pm$0.2	—	—	—	—	—	—	—	—	—	—
Sandstone-1	2.65	114$\pm$7.0	6.6$\pm$0.3	17.0	36.5	54	62	54	63	53	60	4.1	26
Sandstone-2	2.67	174$\pm$10.0	11.6$\pm$0.4	28.0	62.2	59	65	59	65	57	63	2.4	30
Sandstone-3	2.67	87$\pm$4.0	8.3$\pm$0.3	33.3	55.0	53	64	53	65	52	59	1.6	—
Siltstone	2.65	58$\pm$3.0	5.3$\pm$0.2	30.0	48.8	48	57	48	58	48	55	2.9	28
Limestone	2.72	121$\pm$7.0	7.8$\pm$0.3	57.0	37.9	59	64	60	66	55	62	1.4	30
Tuff 1	1.49	10$\pm$0.5	0.9$\pm$0.01	1.1	3.8	—	—	—	—	28	34	—	32
Tuff 2	1.70	11$\pm$0.4	1.2$\pm$0.01	1.4	5.2	50	52	51	56	42	45	—	29
Tuff 3	1.80	27$\pm$0.6	2.6$\pm$0.02	2.4	7.5	38	53	38	54	31	40	—	30
Tuff 4	1.71	14$\pm$0.5	1.5$\pm$0.1	1.6	5.2	45	55	46	56	39	42	—	27
Tuff 5	1.71	19$\pm$0.6	2.3$\pm$0.02	1.3	6.1	43	62	43	61	38	58	—	30
Tuff 6	1.49	6$\pm$0.2	0.2$\pm$0.01	0.4	2.8	19	28	23	32	14	13	—	32

(*): (46–50% Cr_2O_3), (**): (42–46% Cr_2O_3), (***): (20–25% Cr_2O_3), $\pm$SD = standard deviation, γ = density, UCS = uniaxial compressive strength, BTS = Brazilian tensile strength, E_{sta} = static elasticity modulus, E_{dyn} = dynamic elasticity modulus, N24 = N24–type schmidt hammer, L9 = L9-type schmidt hammer, T1, T2, T3 = test procedures as explained in the text, CAI = Cerchar abrasivity index, Ψ = tool-rock friction angle.

cut with a sawing machine and ground to provide smooth ends. In the tests, the PUNDIT™ instrument and two transducers (a transmitter and a receiver) having a frequency of 54 kHz are used. The time difference measured on the oscilloscope between the direct pulse and its arrival after traveling through the sample allows determination of the velocities in the rock and dynamic elasticity modulus.

3.5. Schmidt hammer tests

The following test procedures described by Hucka [88], Poole and Farmer [89], Brown [90] and Ayday and Goktan [91] are used to estimate Schmidt Hammer rebound values:

- Test Procedure 1: Taking the peak rebound value from five continuous impacts at a point and averaging the peaks of the three sets of tests conducted at three separate points.
- Test Procedure 2: Taking the peak rebound value from 10 continuous impacts at a point and averaging the peaks of the three sets of test conducted at three separate points.
- Test Procedure 3: Recording 20 rebound values from single impacts separated by at least a plunger diameter and averaging the upper 10 values.

The correlation between three sets of test procedure was explained in detail by Bilgin et al. [92].

3.6. Density

Trimmed core samples are used in the determination of natural density. The specimen volume is calculated from an average of several caliper readings and the weight of specimen is determined using a sensitive balance. The natural density values are obtained from the ratio of the specimen mass to specimen volume.

3.7. Friction angle between cutting tip material and the rock

A flat surfaced tungsten carbide material with cobalt content of 8% and grain size of 12 μM is used to find the friction angle between freshly saw rock surface and cutting tool material using elementary equipment with different normal loads as explained by Bilgin [12].

3.8. Cerchar abrasivity test

Cerchar abrasivity tests are performed according to test procedures described by West [93] for determining the cutter wear and cost rates due to rock abrasivity. The tests are performed on freshly broken rock surfaces, which are free of weathering effects. The remnant pieces from Brazilian tensile strength tests are used for this purpose.

4. Rock cutting tests

General schematic view of the full-scale cutting rig used in the experiments is presented in Fig. 3. The rig was designed and manufactured within the NATO-TU Excavation Project with sizeable technical help of the Earth Mechanics Institute of Colorado School of Mines [94]. It can accommodate block rock samples up to $0.7 \times 0.7 \times 1\,m^3$ in size. A high-quality aircraft aluminum block equipped with strain gauges is used as a dynamometer to record thrust forces up to 50 tonnes. A data acquisition system is used to record the cutter forces in three perpendicular directions. The data acquisition card includes eight independent channels and monitors the excitation voltage ranging from 0 to 10 V. Data sampling (recording) rate is adjustable up to 50,000 Hz. The hydraulic cylinders can move the sample box in which the rock sample is cast in concrete to eliminate pre-failure of the specimen. The cutter is fixed with a tool holder directly to the dynamometer. The depth of cut is adjusted with hydraulic cylinders as seen in Fig. 3. However, the depth of cut is kept constant using a mechanical device to fix the adjusted depth of cut, which is measured with a depth gauge, at each cut. The surface of rock block, before each set of cut, is trimmed with conical picks used in cutting experiments to have a representative surface. Each cut is replicated at least 3 or 4 times.

The entire test is carried out with an S-35/80 H conical pick manufactured by Sandvik. It has a gauge of 80 mm, flange diameter of 64 mm, shank diameter of 35 mm, tip diameter of 22 mm and primary tip angle of 80°. The constant conditions throughout the testing programme

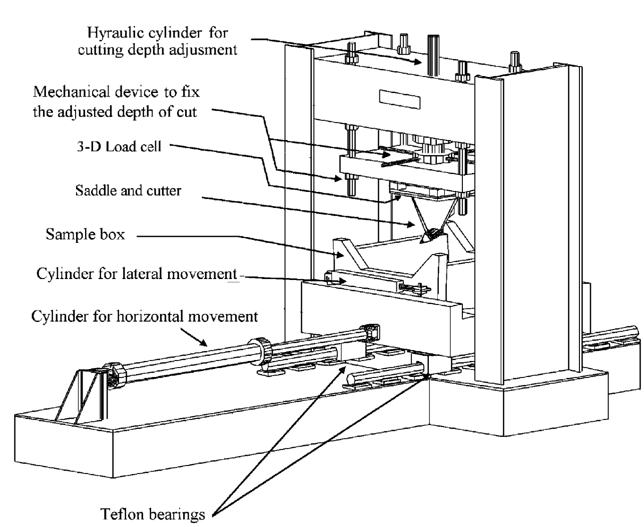

Fig. 3. Schematic drawing of linear cutting machine.

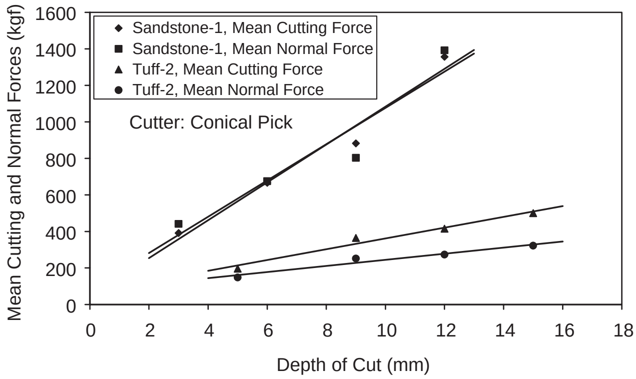

Fig. 4. Relation between cutter forces and depth of cut for Sandstone-1 and Tuff-2.

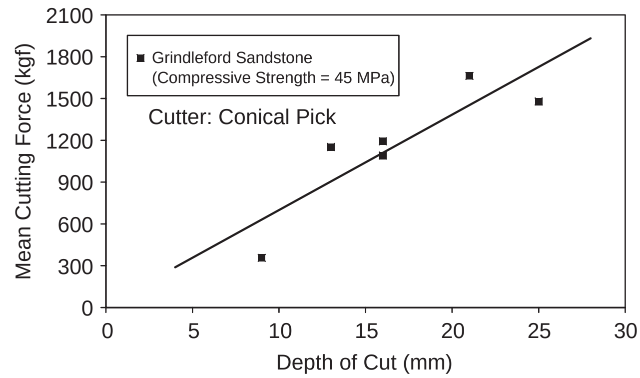

Fig. 6. Relation between cutter forces and depth of cut (after Hurt and Laidlaw 1979 [37]).

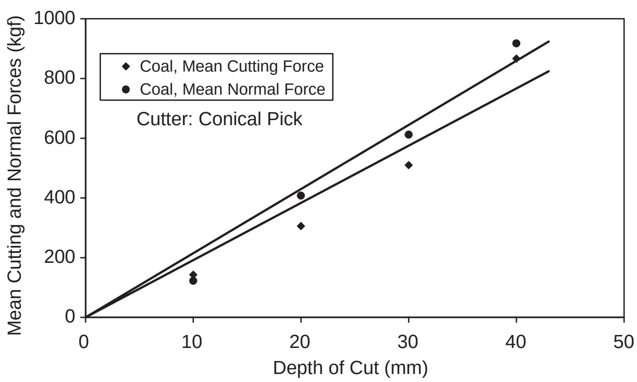

Fig. 5. Relation between cutter forces and depth of cut during shearing of a continuous miner (after Roxborough et al. 1981 [45]).

are attack angle of 55°, cutting speed of 12.7 cm/s and skew and tilt angles of 0°. The data sampling rate is 2000 Hz. The depth of cut is changed usually from 3 to 10 mm and in some cases from 5 to 10 mm. The line spacing is adjusted according to the depth of cut to obtain optimum groove interaction. Peak and mean forces (cutting and normal forces) and yield are recorded in each case. Specific energy is obtained by dividing mean cutting force to yield. Yield is defined as the volume of rock obtained per unit cutting length. The relationship between cutter forces and depth of cut is linear within the limits of depth of cut (from 3 to 10 mm) used for all the rocks tested. Typical linear relationships between depth of cut and cutter forces for Sandstone-1 and Tuff-2 are shown in Fig. 4. The linearity between depth of cut and cutter forces is supported by the previous results of Roxborough [45] and Hurt and Laidlaw [37] as seen in Figs. 5 and 6. This linearity enables the defining of a cutting performance parameter of force to depth of cut ratio (kgf/mm) for each experimental rock, which is used in statistical analysis. Cutting test results for relieved and unrelieved cutting modes as explained in Fig. 2 are presented in Table 2. Relieved cutting test results are for optimum cutting conditions at 9 mm depth of cut. The reason for using 9 mm of depth of cut is that the variation of specific energy versus depth of cut in unrelieved cutting tests usually approaches asymptotically to a minimum level at depth of cut values greater than 9–10 mm for the tested rocks; after this value of depth of cut (d_{opt} as seen in Fig. 2) the specific energy does not change much.

5. Estimation of cutter forces and specific energy from rock properties

The relationships between cutter force to cutting depth ratio and rock properties are presented in Figs. 7–26 for relieved and unrelieved cutting modes. The predictor equations are summarized in Table 3. The best correlations are obtained for uniaxial compressive and tensile strength values suggesting that these are the most important rock properties affecting the performance of conical picks. The third dominant rock property is found to be Schmidt hammer rebound value obtained from N-24-type hammer using Test Procedure 3. Static and dynamic elasticity modulus values yield lower correlations. The predictor equations given in Table 3 may enable any engineer to calculate tool forces from rock properties within acceptable statistical limits.

The published cutting theories for conical picks are of Evans's [9,10], Evans' theory modified by Goktan [18] and Roxborough's [19], which are presented in Eqs. (2), (3) and (4). These theories are only valid for estimating cutting forces in unrelieved cutting mode. However, normal force controlling the depth of cut is a vital factor governing the efficiency of cutting process, since cutting efficiency or specific energy is directly related to depth of cut. In an excavation process, each cut is affected by the adjacent relieving cut as explained in Fig. 2. The tool forces in relieved cutting are lower than those in unrelieved cutting. Table 3 is important in a way that it includes the predictor equations for specific energy,

Table 2
Cutting characteristics of the rocks tested

Rock name	Unrelieved cutting ± SD (mean at different d values)				Relieved cutting ± SD (at optimum (s/d) ratio)					
	FC/d (kgf/mm)	F′C FC	FN/d (kgf/mm)	F′N FN	FC/d (kgf/mm)	F′C FC	FN/d (kgf/mm)	F′N FN	Optimum (s/d) ratio	SE_{opt} (kwh/m^3)
High-grade chromite	54 ± 2	2.69 ± 0.16	41 ± 8	2.50 ± 0.15	40 ± 6	3.62 ± 0.41	27 ± 6	3.29 ± 0.43	3	3.9 ± 0.8
Medium-grade chromite	81 ± 17	2.90 ± 0.07	64 ± 4	2.54 ± 0.09	52 ± 11	2.84 ± 0.31	38 ± 10	2.64 ± 0.26	2	6.4 ± 1.3
Low-grade chromite	69 ± 7	2.59 ± 0.20	60 ± 5	2.29 ± 0.31	51 ± 7	3.15 ± 0.77	40 ± 5	2.89 ± 0.71	3	5.0 ± 1.3
Copper ore (yellow)	43 ± 11	2.93 ± 0.35	30 ± 7	2.85 ± 0.43	40 ± 5	3.1 ± 0.40	26 ± 4	3.1 ± 0.31	4	3.7 ± 0.6
Copper ore (black)	72 ± 17	2.96 ± 0.27	77 ± 13	2.89 ± 0.38	81 ± 7	3.0 ± 0.35	77 ± 5	3.0 ± 0.38	4	9.2 ± 0.9
Harsburgite	104 ± 3	2.87 ± 0.07	115 ± 13	2.31 ± 0.02	101 ± 10	2.89 ± 0.38	105 ± 11	2.43 ± 0.32	5	8.4 ± 2.0
Serpantinite	69 ± 14	2.75 ± 0.13	78 ± 19	2.42 ± 0.40	49 ± 6	3.20 ± 0.50	54 ± 7	2.66 ± 0.13	3	6.2 ± 1.3
Trona	37 ± 13	2.86 ± 0.09	57 ± 20	2.22 ± 0.08	25 ± 5	4.21 ± 0.85	33 ± 6	2.98 ± 0.36	3	2.7 ± 0.6
Anhydrite	75 ± 11	2.87 ± 0.24	85 ± 16	2.70 ± 0.21	56 ± 6	2.4 ± 0.25	57 ± 6	2.0 ± 0.10	5	3.8 ± 0.5
Selestite	43 ± 7	2.82 ± 0.23	33 ± 3	2.63 ± 0.21	33 ± 4	3.1 ± 0.32	25 ± 4	2.8 ± 0.12	3	3.0 ± 0.4
Jips	36 ± 7	2.25 ± 0.13	23 ± 9	2.06 ± 0.17	23 ± 2	2.3 ± 0.28	15 ± 3	2.3 ± 0.13	3	3.4 ± 0.5
Sandstone-1	113 ± 16	2.75 ± 0.43	116 ± 29	2.30 ± 0.39	89 ± 5	2.96 ± 0.10	90 ± 7	2.30 ± 0.90	2	12.6 ± 1.2
Sandstone-2	154 ± 30	2.66 ± 0.36	199 ± 18	2.30 ± 0.15	104 ± 8	3.01 ± 0.24	128 ± 14	2.27 ± 0.20	2	15.4 ± 1.1
Sandstone-3	66 ± 7	2.43 ± 0.06	69 ± 7	2.05 ± 0.11	54 ± 5	2.87 ± 0.31	46 ± 5	2.40 ± 0.19	3	5.4 ± 0.5
Siltstone	105 ± 17	2.99 ± 0.52	132 ± 37	2.51 ± 0.24	70 ± 6	3.40 ± 1.62	80 ± 15	2.80 ± 1.07	3	9.6 ± 0.7
Limestone	129 ± 5	2.86 ± 0.15	229 ± 23	2.03 ± 0.14	129 ± 5	2.83 ± 0.07	192 ± 8	2.11 ± 0.05	5	12.0 ± 1.4
Tuff 1	15 ± 2	2.6 ± 0.3	10 ± 1.7	2.3 ± 0.3	7 ± 0.6	2.3 ± 0.4	4 ± 0.3	2.0 ± 0.2	3	1.6 ± 0.02
Tuff 2	37 ± 3	3.5 ± 0.2	25 ± 3	3.2 ± 0.2	24 ± 1.4	4.5 ± 0.5	16 ± 0.9	4.0 ± 0.3	3	2.7 ± 0.03
Tuff 3	29 ± 3	2.6 ± 0.1	20 ± 0.1	2.5 ± 0.2	24 ± 1.3	2.9 ± 0.3	17 ± 0.8	2.6 ± 0.3	3	2.2 ± 0.06
Tuff 4	22 ± 2	2.9 ± 0.1	15 ± 0.8	2.9 ± 0.3	18 ± 0.5	3.6 ± 0.6	12 ± 0.4	3.0 ± 0.1	3	2.4 ± 0.02
Tuff 5	31 ± 3	2.6 ± 0.1	20 ± 1	2.3 ± 0.2	23 ± 0.7	3.0 ± 0.5	14 ± 0.6	2.6 ± 0.1	3	2.1 ± 0.02
Tuff 6	12 ± 2	2.2 ± 0.1	7 ± 0.8	2.1 ± 0.2	11 ± 0.4	2.4 ± 0.2	6 ± 0.2	2.1 ± 0.2	3	1.3 ± 0.01

s = line spacing, d = depth of cut, FC = mean cutting force, F′C = maximum cutting force, FN = mean normal force, F′N = maximum normal force, SE_{opt} = specific energy obtained at optimum cutting conditions, ±SD = standard deviation.

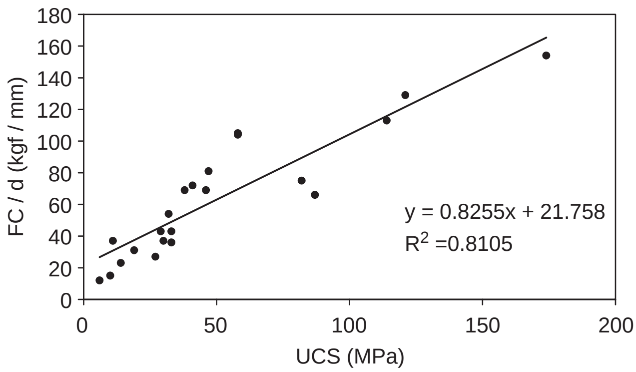

Fig. 7. Relation between mean cutting force to depth of cut ratio and uniaxial compressive strength for unrelieved cutting mode.

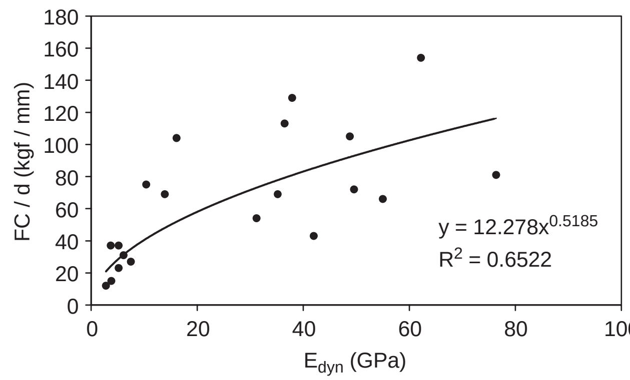

Fig. 10. Relation between mean cutting force to depth of cut ratio and dynamic elasticity modulus for unrelieved cutting mode.

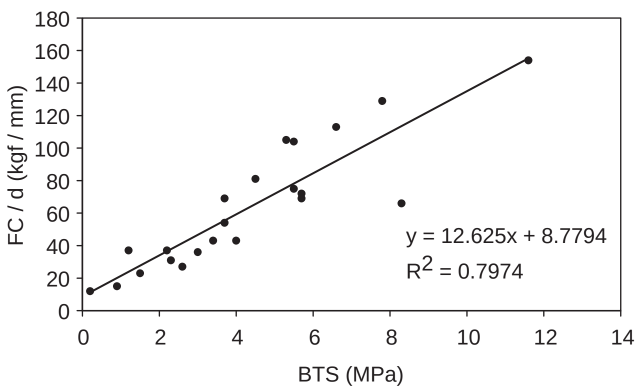

Fig. 8. Relation between mean cutting force to depth of cut ratio and Brazilian tensile strength for unrelieved cutting mode.

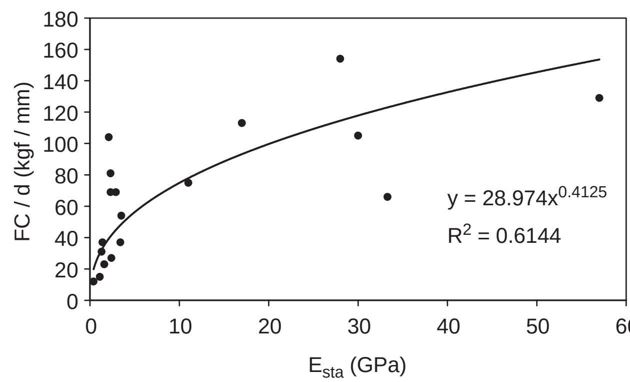

Fig. 11. Relation between mean cutting force to depth of cut ratio and static elasticity modulus for unrelieved cutting mode.

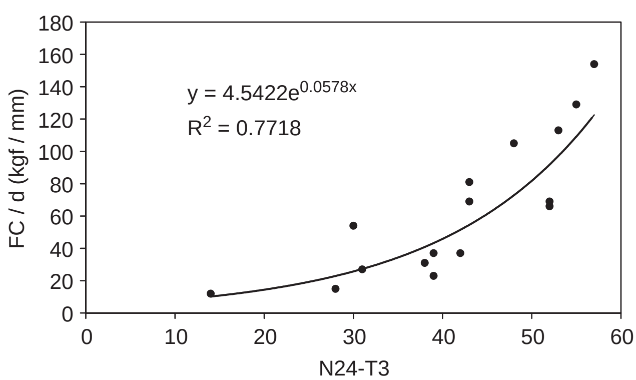

Fig. 9. Relation between mean cutting force to depth of cut ratio and N24-type Schmidt hammer rebound value obtained using test procedure-3 for unrelieved cutting mode.

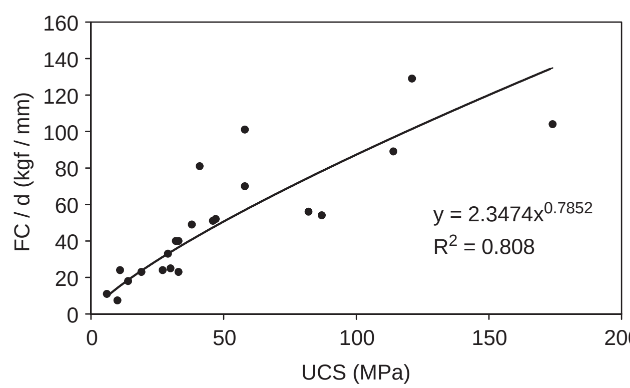

Fig. 12. Relation between mean cutting force to depth of cut ratio and uniaxial compressive strength for relieved cutting mode at optimum cutter spacing to depth of cut ratio.

normal forces and cutting forces in relieved mode, which are not possible to estimate theoretically.

It should also be noted that the predictor equations given in Table 3 are valid for a cutter with 80° tip angle. If the tip angle is different from 80°, then a correction is required. Tip angles are usually manufactured between 60° (with softer rocks) and 90° (with stronger rocks) and 75–80° are the most widely used tip angles. Experimental studies performed by United States of Bureau of Mines indicated that cutting force difference was around 70% more with a 90° tip angle than a 60° tip angle, while around 3 times more in normal force [95]. It is seen that Evans' theoretical cutting force in Eq. (2) includes tip angle (φ) and yields around 50% difference between 60°

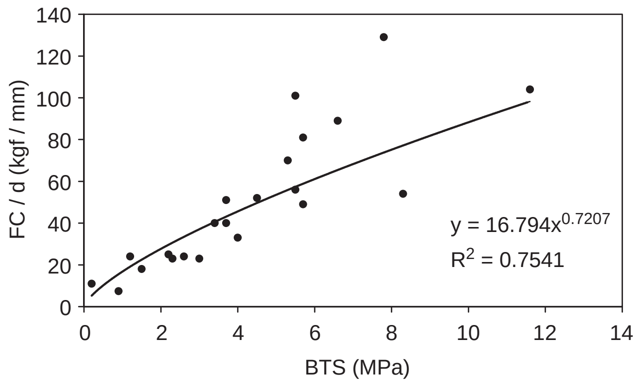

Fig. 13. Relation between mean cutting force to depth of cut ratio and Brazilian tensile strength for relieved cutting mode at optimum cutter spacing to depth of cut ratio.

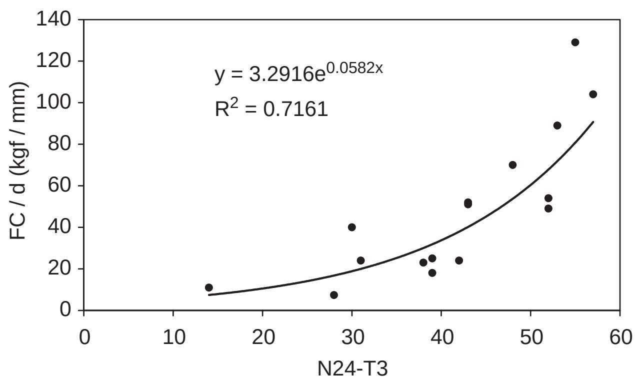

Fig. 14. Relation between mean cutting force to depth of cut ratio and N24-type Schmidt hammer rebound value obtained using test procedure-3 for relieved cutting mode at optimum cutter spacing to depth of cut ratio.

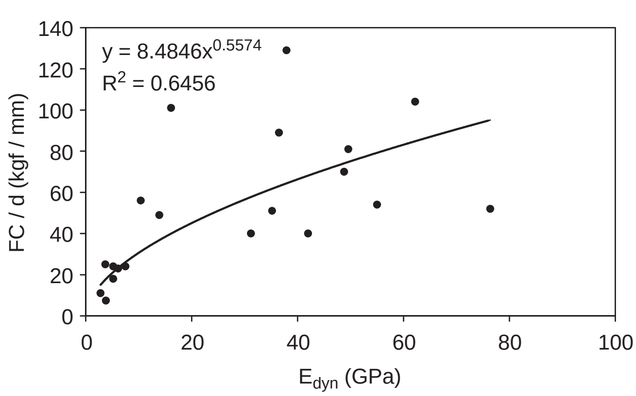

Fig. 15. Relation between mean cutting force to depth of cut ratio and dynamic elasticity modulus for relieved cutting mode at optimum cutter spacing to depth of cut ratio.

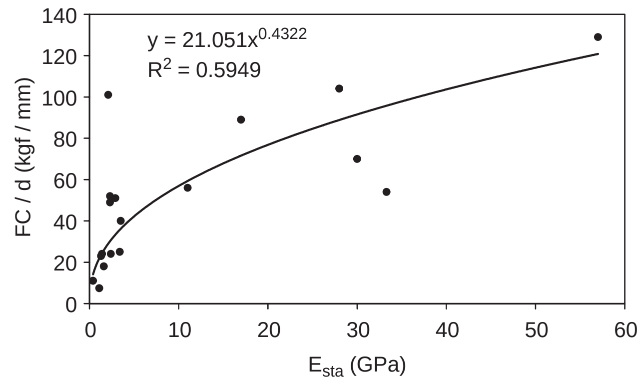

Fig. 16. Relation between mean cutting force to depth of cut ratio and static elasticity modulus for relieved cutting mode at optimum cutter spacing to depth of cut ratio.

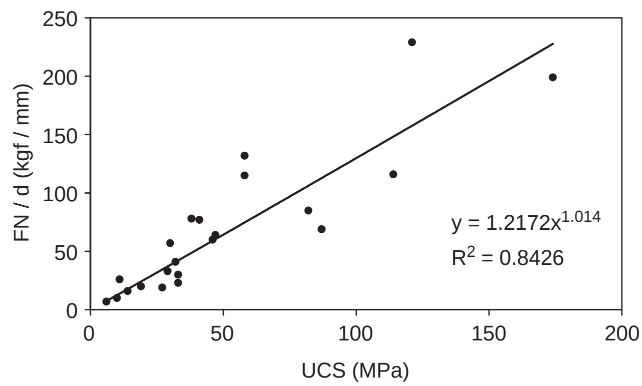

Fig. 17. Relation between mean normal force to depth of cut ratio and uniaxial compressive strength for unrelieved cutting mode.

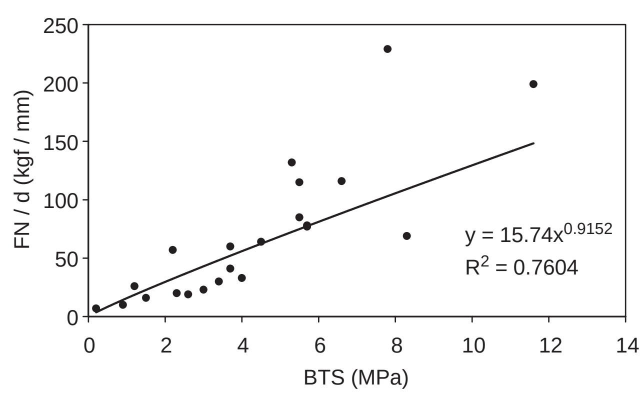

Fig. 18. Relation between mean normal force to depth of cut ratio and Brazilian tensile strength for unrelieved cutting mode.

and 90° tip angles for conical picks. Therefore, the parameter "$1/\mathrm{Cos}^2(\varphi/2)$" may be used as a tip angle correction factor for cutting force. However, the correction for normal force requires further studies.

Designing or practicing engineers are also interested in the ratio of peak to mean forces since this ratio is an important factor affecting the vibration of a cutting head and the breakdown of the mechanical parts. The experimental results indicate that this ratio is not affected by the rock properties. The ratio of peak cutting force to mean cutting force and the ratio of peak normal force to mean normal force are found to be

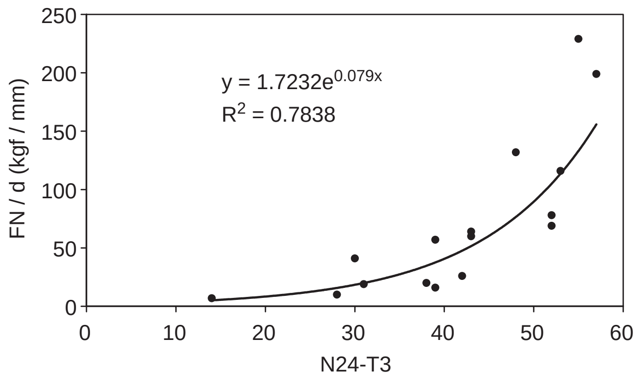

Fig. 19. Relation between mean normal force to depth of cut ratio and N24-type Schmidt hammer rebound value obtained using test procedure-3 for unrelieved cutting mode.

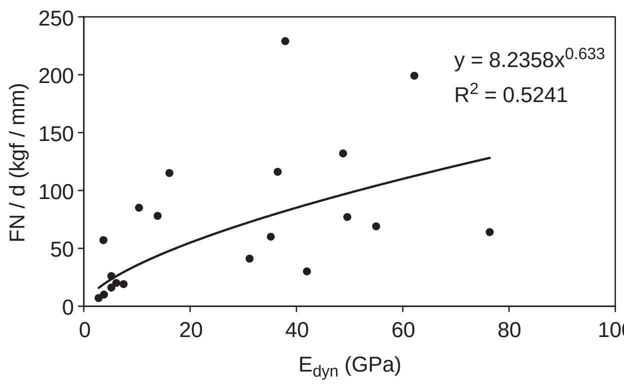

Fig. 20. Relation between mean normal force to depth of cut ratio and dynamic elasticity modulus for unrelieved cutting mode.

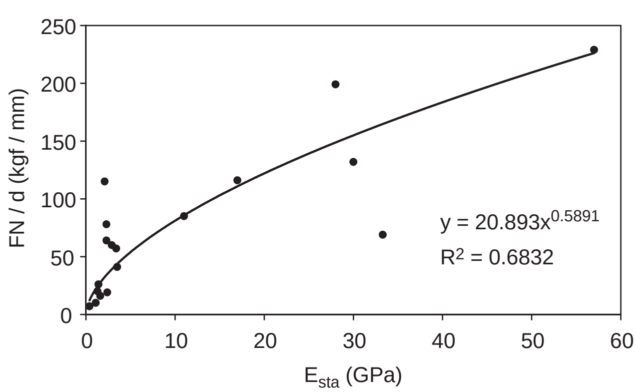

Fig. 21. Relation between mean normal force to depth of cut ratio and static elasticity modulus for unrelieved cutting mode.

2.69 ± 0.32SD (standard deviation) and 2.39 ± 0.33SD for unrelieved cutting mode and 3.07 ± 0.55SD and 2.64 ± 0.49SD for relieved cutting mode, respectively. The values measured in relieved cutting mode are higher than those in unrelieved mode. This is expected, since larger chips are obtained in relieved cutting increasing peak forces.

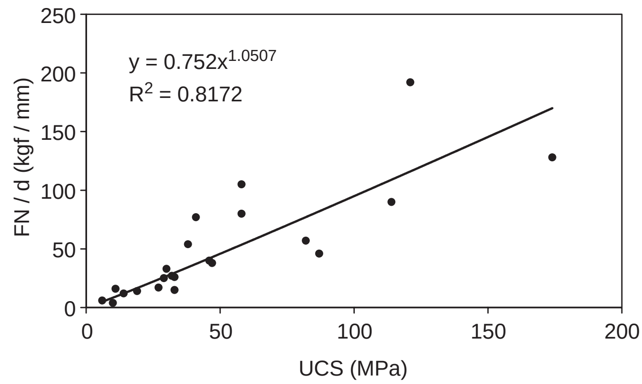

Fig. 22. Relation between mean normal force to depth of cut ratio and uniaxial compressive strength for relieved cutting mode at optimum cutter spacing to depth of cut ratio.

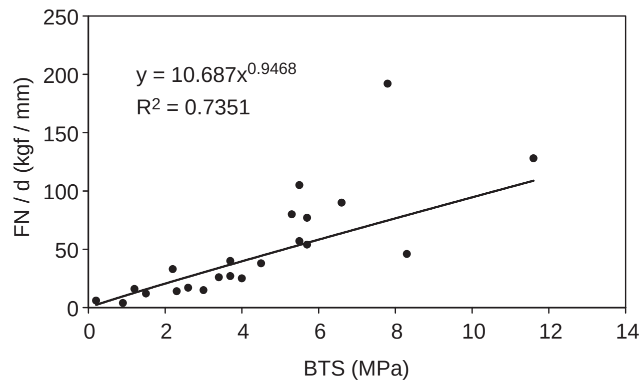

Fig. 23. Relation between mean normal force to depth of cut ratio and Brazilian tensile strength for relieved cutting mode at optimum cutter spacing to depth of cut ratio.

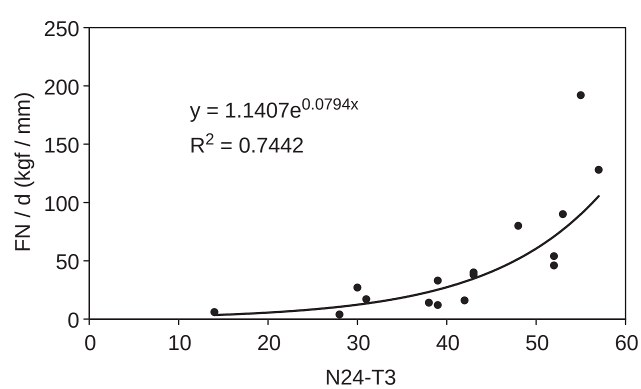

Fig. 24. Relation between mean normal force to depth of cut ratio and N24-type Schmidt hammer rebound value obtained using test procedure-3 for relieved cutting mode at optimum cutter spacing to depth of cut ratio.

The estimation of optimum specific energy is important in predicting cutting rates of excavation machines as explained in Eq. (11). As seen in Figs. 27–31, specific energy is best predicted from uniaxial compressive strength and tensile strength verifying some

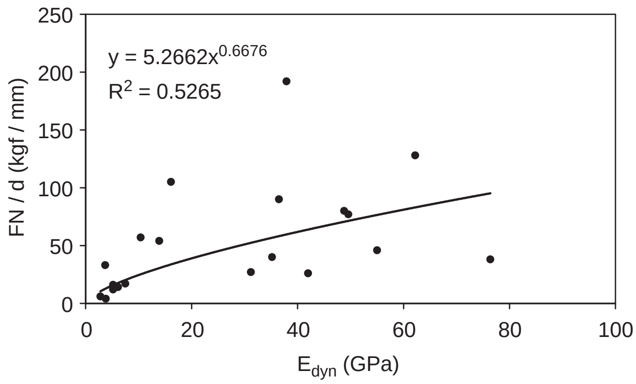

Fig. 25. Relation between mean normal force to depth of cut ratio and dynamic elasticity modulus for relieved cutting mode at optimum cutter spacing to depth of cut ratio.

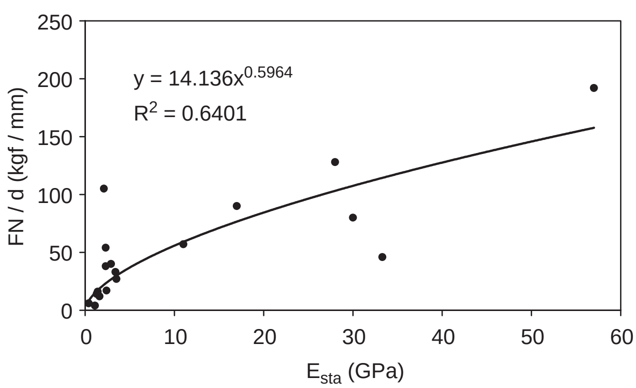

Fig. 26. Relation between mean normal force to depth of cut ratio and static elasticity modulus for relieved cutting mode at optimum cutter spacing to depth of cut ratio.

of the previously published results [86]. Moderate correlations are obtained for Schmidt hammer rebound values for N-24-type hammer using Test Procedure 3 and static and dynamic elasticity modulus.

Cutter spacing to depth of cut ratio (s/d) is the key factor in obtaining optimum specific energy, hence the most efficient cutting conditions. Bearing in mind that cutter spacing on a cutting drum is fixed by the machine manufacturer, the only way to excavate in optimum condition is to apply the operational parameters such as arcing force giving the desired depth of cut. The optimum s/d ratio is found varying between 2 and 5 for the 22 rocks tested. However, a direct relationship between rock properties and optimum s/d ratio could not be found within the limits of this research programme. It is strictly recommended that this aspect of rock cutting mechanics needs further research.

6. Comparison of experimental and theoretical values

Evans suggests that uniaxial compressive strength and tensile strength governs the failure in rock cutting with

Table 3
Regression equations to predict cutter performance based on rock properties

Unrelieved cutting		Relieved cutting (at optimum s/d)	
Regression equation	R^2	Regression equation	R^2
$FC/d = 0.826\ \sigma_c + 21.76$	0.810	$FC/d = 2.347\ \sigma_c^{0.785}$	0.808
$FC/d = 12.625\ \sigma_t + 8.78$	0.797	$FC/d = 16.794\ \sigma_t^{0.721}$	0.754
$FC/d = 4.542\ e^{0.058\ SH}$	0.772	$FC/d = 3.292\ e^{0.058\ SH}$	0.716
$FC/d = 12.278\ E_{dyn}^{0.519}$	0.652	$FC/d = 8.485\ E_{dyn}^{0.557}$	0.646
$FC/d = 28.974\ E_{sta}^{0.413}$	0.614	$FC/d = 21.051\ E_{sta}^{0.432}$	0.595
$FN/d = 1.217\ \sigma_c^{1.014}$	0.843	$FN/d = 0.752\ \sigma_c^{1.051}$	0.817
$FN/d = 15.74\ \sigma_t^{0.915}$	0.760	$FN/d = 10.687\ \sigma_t^{0.947}$	0.735
$FN/d = 1.723\ e^{0.079\ SH}$	0.784	$FN/d = 1.141\ e^{0.079\ SH}$	0.744
$FN/d = 20.893\ E_{sta}^{0.589}$	0.683	$FN/d = 14.136\ E_{sta}^{0.596}$	0.640
$FN/d = 8.236\ E_{dyn}^{0.633}$	0.524	$FN/d = 5.266\ E_{dyn}^{0.668}$	0.527
		$SE_{opt} = 0.083\ \sigma_c + 1.424$	0.760
		$SE_{opt} = 1.259\ \sigma_t + 0.142$	0.743
		$SE_{opt} = 0.3912\ e^{0.058\ SH}$	0.757
		$SE_{opt} = 0.984\ E_{dyn}^{0.542}$	0.682
		$SE_{opt} = 2.424\ E_{sta}^{0.414}$	0.617

FC = mean cutting force in kgf, FN = mean normal force in kgf, d = depth of cut in mm, SE_{opt} = specific energy at optimum cutting condition in kWh/m^3, σ_c = uniaxial compressive strength in MPa, σ_t = brazilian tensile strength in MPa, E_{sta} = static elasticity modulus in GPa, E_{dyn} = dynamic elasticity modulus in GPa, SH = N-24-type schmidt hammer rebound value using test procedure 3 as explained in the text.

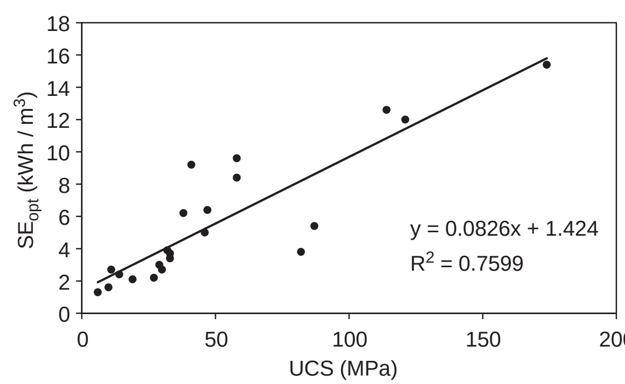

Fig. 27. Relation between optimum specific energy and uniaxial compressive strength.

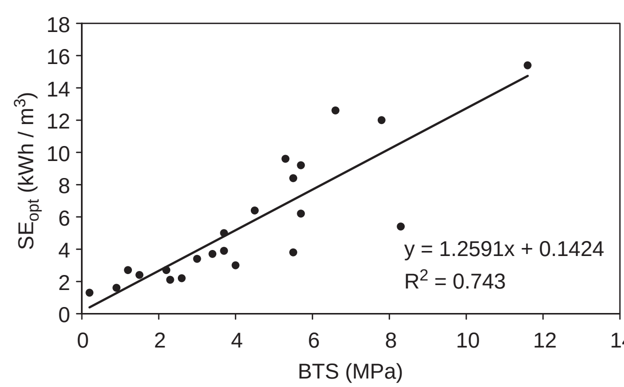

Fig. 28. Relation between optimum specific energy and Brazilian tensile strength.

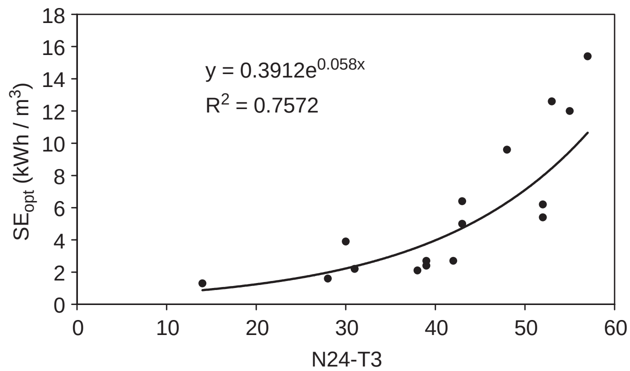

Fig. 29. Relation between optimum specific energy and N24-type Schmidt hammer rebound value obtained using test procedure-3.

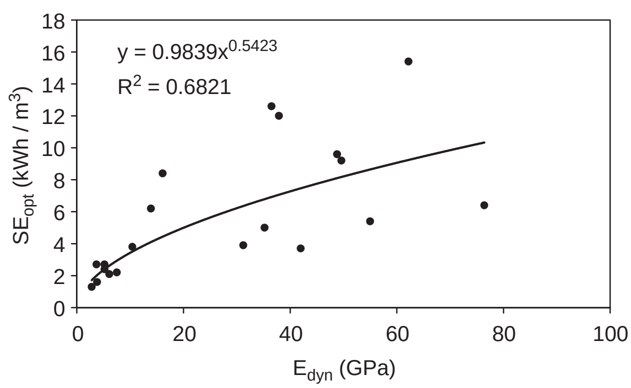

Fig. 30. Relation between optimum specific energy and dynamic elasticity modulus strength.

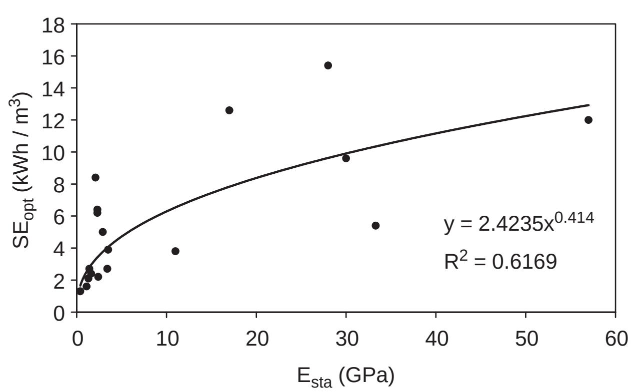

Fig. 31. Relation between optimum specific energy and static elasticity modulus strength.

conical picks [9,10]. However, tensile strength, compressive strength and friction angle between cutting tool and rock are dominant factors in Goktan's and Roxborough's modified theories [18,19]. The rock parameters given in Table 1 are used to estimate theoretical cutting forces based on equations suggested by Evans (Eq. (2)), Goktan (Eq. (3)) and Roxborough (Eq. (4)). Theoretical and experimental values are presented and compared in Table 4.

A statistical analysis is carried out in order to see whether the experimental cutting force values are significantly different from the theoretical cutting force values. One of the most common tests in determining whether one process is different from another is the student t-test, based on comparison between pairs of values. The closer to zero that the total of the difference lines, the more confident that there is no difference between the processes. The test compares the calculated t-value with a tabulated t-value using the null hypothesis. We reject the hypothesis of no difference and conclude that the experimental and theoretical cutting force values are different when the calculated t-value is greater than the tabulated t-value. A confidence level of 95% is chosen for the current analysis.

The results of statistical analysis are summarized in Table 5. As it is seen, the calculated t-value for mean experimental force values is 0.31 and for Goktan's theoretical cutting force values is 2.11 at 5 mm depth of cut. Since the tabulated t-value is greater than the calculated t-value, the mean experimental force values are not significantly different from the Goktan's theoretical cutting force values. The other comparisons between the experimental and theoretical cutting force values are significantly different, since the calculated t-values are greater than the tabulated t-values.

Specific energy might be formulated as given in Eq. (6) using compressive strength and elasticity modulus of rock. Bearing in mind that cutter parameters and operational parameters, such as tip angle, depth of cut and cutter spacing also influence optimum specific energy. Eq. (6) is proved to be valid as a guide in estimating cutting rate of mechanical excavators as suggested by Farmer and Garrity [57], Pool [58], Krupa, Sekula and co-workers [59–62]. As presented in Fig. 32, there is a close relationship between predicted and experimental specific energy values if the rocks are grouped according to their static elasticity modulus with the cutoff value of 15 GPa. This indicates that static elasticity modulus has an important influence on cutting characteristics of rocks.

7. Conclusions

Conical picks are the essential cutting tools used on many mechanical excavators, so it is important to understand their cutting behavior in different rock formations in order to design efficient cutting systems, work in operational parameters and estimate cutting performance of excavating machines. The results of experimental studies indicate that uniaxial compressive strength and tensile strength are best correlated with the measured cutting and normal force values. Predictor

Table 4
Comparison of experimental and theoretical cutting forces for unrelieved cutting mode

Rock name	Depth of cut = 5 mm					Depth of cut = 9 mm				
	Experimental cutting force		Theoretical cutting force			Experimental cutting force		Theoretical cutting force		
	Mean (kgf)	Peak (Kgf)	Evans (kgf)	Goktan (kgf)	Roxborough (kgf)	Mean (kgf)	Peak (kgf)	Evans (kgf)	Goktan (kgf)	Roxborough (kgf)
High-grade chromite	279	716	92	252	107	530	1483	366	1008	348
Medium-grade chromite	347	1021	92	307	121	931	2649	369	1226	393
Low-grade chromite	319	871	64	283	97	663	1624	255	1131	313
Copper ore (yellow)	170	440	75	492	121	509	1507	300	1966	393
Copper ore (black)	270	733	170	524	194	908	2582	679	2097	627
Harsburgite	531	1497	112	336	141	922	2691	447	1343	456
Serpantinite	295	785	183	411	182	710	2015	732	1644	589
Trona	139	388	35	178	56	420	1226	138	714	182
Anhydrite	338	1252	79	—	—	519	1630	316	—	—
Selestite	150	474	118	—	—	343	907	473	—	—
Jips	401	872	58	—	—	338	653	234	—	—
Sandstone-1	758	1969	82	425	133	992	2952	327	1702	432
Sandstone-2	820	2325	166	941	282	1686	4810	662	3763	913
Sandstone-3	379	909	170	—	—	655	1592	678	—	—
Siltstone	741	2304	104	382	143	843	3200	415	1528	464
Limestone	746	2151	108	633	186	1217	3285	431	2531	603
Tuff 1	74	205	17	83	27	161	402	69	331	86
Tuff 2	196	708	28	92	36	387	1184	112	367	115
Tuff 3	125	377	54	211	75	274	722	214	844	244
Tuff 4	93	283	34	102	42	248	730	138	409	137
Tuff 5	137	344	60	187	72	299	735	238	746	234
Tuff 6	47	133	1.4	18	3	104	218	5.7	74	11

Table 5
Statistical analysis of experimental and theoretical cutting forces

Depth of cut	Data pairs	Calculated t-value	Tabulated t-value
5 mm	Mean-Evans	5.31 (df* = 21)	2.08
	Mean-Goktan	0.31 (df = 17)	2.11
	Mean-Roxborough	4.35 (df = 17)	2.11
	Peak-Evans	6.04 (df = 21)	2.08
	Peak-Goktan	4.44 (df = 17)	2.11
	Peak-Roxborough	5.07 (df = 17)	2.11
9 mm	Mean-Evans	4.52 (df = 21)	2.08
	Mean-Goktan	-4.90 (df = 17)	2.11
	Mean-Roxborough	-6.39 (df = 17)	2.11
	Peak-Evans	6.48 (df = 21)	2.08
	Peak-Goktan	4.29 (df = 17)	2.11
	Peak-Roxborough	6.34 (df = 17)	2.11

(*) df = degree of freedom.

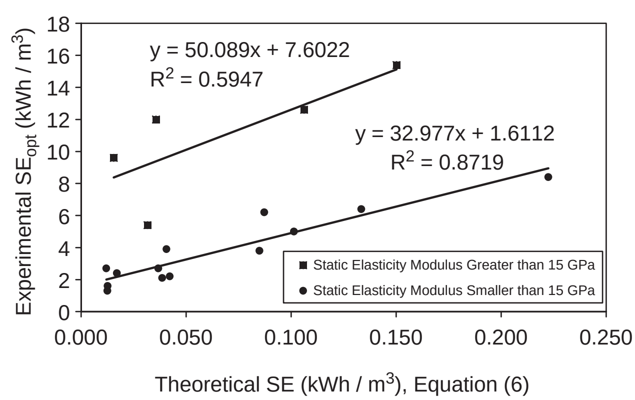

Fig. 32. Relation between theoretical specific energy and optimum specific energy obtained from full-scale linear cutting tests.

equations are given for force and optimum specific energy values. These equations may enable engineers to calculate tool forces for relieved and unrelieved conditions from rock properties within acceptable statistical limits. The third dominant rock property is found to be Schmidt hammer rebound value obtained from N-24-type hammer using Test Procedure 3 as explained in the paper. The ratio of peak to mean forces is found to change between 2.4 and 3.1 and is not affected by the rock properties. Cutter spacing to depth of cut ratio is the key factor in obtaining the most efficient cutting condition; this ratio is found to change between 2 and 5. Any statistical relationship between rock properties and optimum cutter spacing to depth of cut ratio could not be found within the limits of this research and further research work is recommended in this area.

Theoretical studies always serve as a guide to practicing engineers to eliminate the large cost of trial and error approaches. The student-t test is carried out in order to see whether the experimental cutter force values are significantly different from the theoretical cutting force values obtained from the Evans' [9,10], Goktan's [18] and Roxborough's [19] predictor equations. It is observed that the friction angle between cutting tool and rock plays an important role in estimating cutting force, and Goktan's modified Evans' cutting theory gives the best results only for 5 mm depth of cut. The parameter "$1/\mathrm{Cos}^2(\varphi/2)$" used in Evans' theoretical cutting force equation may be used as a tip angle correction factor, while further studies are required for normal force correction factor. These results indicate that there is definite need in improving cutting theories in rock cutting mechanics science.

Theoretical specific energy as formulated by Hughes [47] and Mellor [48] is correlated well with experimental optimum specific energy values under specific conditions. This suggests that theoretical specific energy may be used as a guide in estimating the performance of the cutting machines.

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